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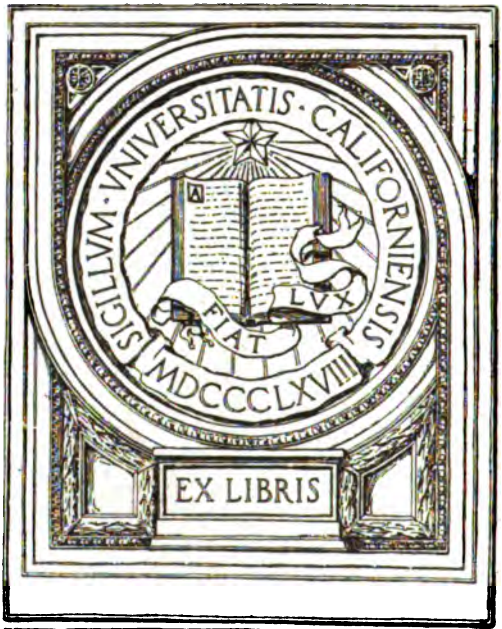

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# ELEMENTS

OF

# METALLURGY.

## A PRACTICAL TREATISE ON THE ART OF EXTRACTING METALS FROM THEIR ORES.

BY

**J. ARTHUR PHILLIPS,**

F.R.S., F.G.S., M. INST. C.E., F.C.S.,

ANCIEN ÉLÈVE DE L'ÉCOLE DES MINES.

*NEW EDITION, REVISED AND ENLARGED*

BY

THE AUTHOR AND H. BAUERMAN, F.G.S.

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# PREFACE TO THE SECOND EDITION.

THE former issue of this work having been exhausted, the present revised edition was commenced under the conditions indicated on the title-page, and had proceeded far towards completion when, to the extreme regret of a large circle of personal and professional friends, the Author died suddenly on the morning of January 5, 1887, having been engaged upon the work of revision to within a very few hours of his lamented decease.

The whole of the manuscript was at that time in the printer's hands, and the work has been completed with no more alteration than would, in the opinion of his coadjutor, have been adopted by the Author himself had he survived to complete his work.

The progress of invention in all branches of metallurgical industry during the last ten years has necessarily led to an enlargement of the volume, which has been increased in the text from 739 to 817 pages, and in the number of illustrations from 205 to 232, although some considerable reductions have been made by the suppression of matter and illustrations either obsolete or unsuited to present conditions contained in the former edition.

The largest alterations have, as might be expected, been made in the section devoted to Iron, but many notable additions will also be found under the heads of Copper, Lead, Silver, and Gold, dealing with the newer developments in processes for the smelting and extraction of these metals from their ores. It was originally intended to have entered at some length upon the subjects of gaseous fuel and firing in their newer developments, but as this could only be done by trenching upon the space required for the more immediate purpose of the book, namely, the description of smelting-processes proper, the intention has been abandoned, although not without some regret.

In the revision of the text, notices have been removed of several processes which have either failed in practice or become obsolete; but those coming within the latter category, that are either of historical interest or form the bases of subsequent developments, have been retained.

LONDON, *April 20*, 1887.

# PREFACE TO THE FIRST EDITION.

THE First Edition of the Author's 'Manual of Metallurgy' appeared in 1852, the second in 1854, and the third in 1858. Up to the last of these dates no other work on this subject had been published in England; but we have since had the excellent volumes of Dr. Percy, and a translation of Professor Kerl's Metallurgy, with additions, by Messrs. Crookes and Röhrig.

A treatise on the 'Metallurgy of Iron,' by W. Truran, is also before the public, as well as an admirable manual on the same subject by H. Bauerman; two small but useful volumes on the Metallurgy of Copper, Lead, and Silver, by Dr. R. H. Lamborn, forming a portion of 'Weale's Rudimentary Series,' and various other works of a somewhat similar character, have from time to time been published within the last fifteen years.

The student has, therefore, within his reach a considerable number of metallurgical books in the English language; but some of them are large and consequently expensive, while others confine themselves to the metallurgy of one of the metals only. No well-illustrated treatise in a single volume has, however, appeared, describing with any considerable detail the metallurgical operations relating to all the principal metals, and at the same time serving as an introduction to the general literature of metallurgy.

The object of the present work is to supply, within moderate limits, such practical information on general principles, and typical processes, as may not only afford a comprehensive view of the subject, but also enable the reader to study with advantage more elaborate treatises and original memoirs. The information generally is, as far as practicable, brought up to the present time. In order to do this, various works on metallurgy, as well as numerous papers dispersed through the different English and foreign scientific journals, have been consulted. Recourse has also been had to notes which, during the last twenty years, have been made by the Author, both at home and abroad. These principally refer to the Metallurgy of Copper, Lead, Silver, and Gold.

In the case of each of the more important metals, the different ores from which they are respectively obtained are enumerated and described, while statistics of their distribution and production in various parts of the world are also added. Much information relative to the distribution of iron ore has been derived from Bauerman's 'Metallurgy of Iron'; in the case of the other metals the statistics have either been obtained from official sources, or compiled from the most trustworthy information available. For returns relative to the production of France and of the United States of America, we are indebted respectively to Professor Daubrée, of the École des Mines, Paris, and to Professor B. Silliman, of Yale College, Connecticut.

A great difficulty experienced in collecting statistics relative to the metal-production of the world arises from the fact that ores raised in one country are frequently subjected to metallurgical treatment in another, and the produce is, in the majority of instances, returned by both.

Allowance for such double returns has, when practicable, been made, but the complete elimination of error arising from this cause would be exceedingly difficult. Sometimes no precise figures have been available, and in such cases estimates founded on the best procurable data have been substituted.

Both the wet and dry methods of assaying are given with considerable detail, and when, as in the case of ironstones, the value of an ore is materially affected by the nature and quantity of the impurities present, the processes employed for their detection and estimation are fully described.

The principal works, in addition to those already mentioned, to which the Author wishes to acknowledge his obligations, are the following: Gruner and Lan, 'Métallurgie du Fer en Angleterre,' published in the *Annales des Mines;* Gruner, 'De l'Acier et de sa Fabrication,' also published in the *Annales des Mines;* Jordan, 'Métallurgie du Fer au pays de Siegen,' published in De Kuyper's *Revue Universelle;* Dana's 'System of Mineralogy,' and Watts's 'Dictionary of Chemistry.' Many other books and papers have necessarily been consulted; but the several sources from which information has been derived will be found duly acknowledged in the text.

Of the illustrations a large number are new, and have been reduced by Mr. W. J. Welch from working drawings, a few only having been retained from the 'Manual of Metallurgy.' Some have been obtained from foreign sources, particularly from drawings published by the Technical Institute of Berlin under the title of 'Sammlung von Zeichnungen für die Hütte.' Others have been reproduced from papers published in the Transactions of various societies; a few have been reduced from Truran's drawings; and some six or eight have, with the kind permission of the Author, been adapted from Percy's 'Metallurgy.'

The woodcuts have been drawn to scale, and a sufficient number of dimensions are generally given to render easy the determination of any others that may be required.

The Author has much pleasure in acknowledging his obligations to Mr. W. Hutchison for information relative to lead-smelting at Couëron, to Professor Ulrich for a description of the process employed at Oker for the extraction of silver and gold from copper by sulphuric acid, and to Ober-Berg- und Hütten-Director Leuschner for drawings of the large blast-furnace at Mansfeld, and for valuable information relative to the various processes employed in that establishment. His thanks are likewise due to Mr. F. W. Rudler for his able assistance during the time this volume has been passing through the press.

CRESSINGTON PARK, AIGBURTH,
*June*, 1874.

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# ELEMENTS OF METALLURGY.

## INTRODUCTION.

**Metallurgy** is the art of extracting metals from their ores and preparing them for the uses of the artizan and manufacturer. A knowledge of the principles involved in the treatment of metalliferous substances for the metals they contain constitutes the science of metallurgy. The various phenomena observed during metallurgical processes relate either to chemistry or to physics. Mechanical appliances are also extensively employed by the metallurgist, and the science of metallurgy is consequently founded on a knowledge of chemistry, physics, and mechanics.

The history of the art dates from the most remote antiquity, and its fundamental principles had been discovered and applied to the wants of mankind, long before the existence of the sciences by the aid of which their operations have since been explained. Tubal-Cain is stated to have been "an instructor of every artificer in brass and iron."[1] In the days of Moses, at least six metals were known, since, in his direction for the purification of the spoils of the Midianites, he says: "Only the gold, and the silver, the brass, the iron, the tin, and the lead, everything that may abide the fire, ye shall make it go through the fire, and it shall be clean."[2]

That silver was at a very early period extracted from ores of lead is apparent from the following passages, which evidently refer to cupellation. "The house of Israel is to me become dross: all they are brass, and tin, and iron, and lead, in the midst of the furnace; they are even the dross of silver."[3] And again, "The bellows are burned, the lead is consumed of the fire; the founder melteth in vain; for the wicked are not plucked away. Reprobate silver shall men call them, because the Lord hath rejected them."[4] Strabo quotes Polybius as speaking of an ore which, after being washed seven times, was melted with lead and became pure silver.

In speaking of gold, which was probably one of the first metals known, Pliny says: "In these parts of the world in which we live, gold mines are found, to say nothing of India, where the ants cast it up out of the ground, or that which the griffins gather in Scythia. The gold with us is procured in three ways. Among the sands of some great rivers, such as the Tagus in Spain, the Po in Italy, Hebrus in Thrace, Pactolus in Asia, and the Indian Ganges, all of which yield gold. Neither is there any gold finer or more perfect, from being thoroughly polished by the rubbing and attrition it meets with in the courses of streams of water. There is another method of obtaining gold, viz., by digging it out of pits which are made for that purpose, or else in caverns and breaches caused by the fall of mountains."[1] He goes on to say, "Other minerals after their extraction require fire for their conversion into metal; but gold, of which we now treat, is gold as soon as it is found." Again, "Neither rust nor canker alters the weight of gold, or affects in any way its quality. Salt and vinegar, though such active solvents, do not make the least impression on it." He states that "with respect to its purification it should be mixed with lead." No mention is made of separating gold from silver, although Pliny observes that all gold contains more or less silver, and adds that when that metal is in the proportion of one-fifth, the alloy is called *electrum*. There were anciently extensive gold mines in Thasos and other Greek islands. Herodotus tells us he had himself seen the mines of Thasos, and that a great mountain had been overturned in searching for the metal.[2]

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[1] Gen. iv. 22.
[2] Num. xxxi. 22, 23.
[3] Ezek. xxii. 18.
[4] Jer. vi. 29, 30.

Gold was employed in Rome for the purpose of fixing artificial teeth more than three centuries before the Christian era, and a law of the Twelve Tables makes exception with regard to such gold, permitting it to be buried with the dead.[3] The remains of numerous mines have been traced by Gmelin, Lepechin, and Pallas, on the southern and eastern borders of the Ural mountains; and in them were found hammers and chisels of copper, as well as various instruments of the same metal, of which the uses are at present unknown. From the absence of any remains of masonry in the neighbourhood, these excavations are inferred to have been made by a nomadic people, probably the Scythians; and from no iron tools having been found in any of them, we may conclude that these operations were carried on before the conquest of Siberia by the Tartars, who effected the subjugation of that part of Asia about 150 years before our era.[4] Hammers made of large stones, to which handles had been attached, were also discovered, together with boars' fangs, with which the gold appears to have been collected, and leathern bags or pockets in which it was preserved. With such imperfect tools, the progress made must necessarily have been slow, and in one instance, after reaching a band of rock, and penetrating it for a short distance, the miners seem to have lost patience and abandoned the works.

Lumps of copper, containing no traces of gold, have also been discovered, although the copper ores of the district are found associated with gold, and it is therefore probable that the ancient people who worked these mines were acquainted with a method of separating gold from copper.

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[1] Pliny, Hist. Nat. xxxiii. 4. [2] Lib. vi. c. 47. [3] Cic. de Leg. ii. 24.
[4] 'Histoire Généalogique des Tartares.'

Smelting was effected in small furnaces made of red bricks. Gmelin found nearly a thousand such furnaces in the eastern parts of Siberia. The height and breadth of these were each about two feet, and the length three. They were furnished with holes in two of their opposite sides, the one for the introduction of a blast, and the other for the escape of the metal and slags. In the neighbourhood of the furnaces were found large quantities of broken pottery, together with numerous heaps of scoriæ, which indicate that operations to a very considerable extent had, at some period, been carried on in the locality.

Gmelin likewise found in the same district the remains of various furnaces which had been employed for the extraction of silver, and remarked that the lead with which it was associated had been thrown away in the scoriæ, whilst the whole of the silver was carefully extracted. By what means this was effected, in this particular case, it is of course now impossible to determine, although it is highly probable that cupellation in some form was resorted to. Diodorus (iii. 14) informs us, that gold was purified by being melted and heated in earthen pots, together with an alloy of tin and lead, to which salt and barley-bran were added; and that the fire was kept up during five successive days. Hippocrates states that gold was melted by a gentle fire, with the addition of salt, nitre and alum, and that the same process was employed for refining silver.

Mercury is first mentioned by Aristotle and Theophrastus under the name of fluid silver (*ἄργυρος χυτός*); but its nature does not appear to have been well understood even four centuries later, since Pliny distinguishes between quicksilver, *argentum vivum*, and the liquid silver, *hydrargyrus*, obtained by the treatment of native cinnabar. The latter he supposes to be a spurious imitation of quicksilver and a fraudulent substitute for it.[1] With regard to the properties of quicksilver, he observes: "So penetrating is this liquor, that there is no metal but it will eat and pass through. It supports everything which may be thrown into it, unless it be gold only, which sinks to the bottom. It is, besides, very useful for the purpose of refining gold; to effect which object that metal mixed with cinders is placed in an earthen pot, and shaken with mercury, which rejects all the impurities mixed with it, but in return takes hold of the gold itself. To expel it from the gold, the mixture is poured on skins, which, on being pressed, allow the mercury to pass through them in drops, whilst the gold remains in all its purity."[2]

The above process differs but little from the methods in general use for the purposes of amalgamation at the present day; but in this case Pliny's description is imperfect, inasmuch as the solid amalgam remaining on the skins would require the separation of the combined mercury, by the aid of heat, before the gold could exist in the pure and fine state described.

Tin and lead appear to have been frequently confounded by the ancients, since their names in Hebrew, Arabic, Greek and Latin are often indifferently used.[1] The Greeks, when they would distinguish the two metals, called tin κασσίτερος, and lead μόλυβδος. Pliny appears to have regarded them as two varieties of the same metal, as he describes them under the titles of white lead and black, and states that *plumbum candidum*, called by the Greeks κασσίτερος, was more valuable, and commanded a higher price than the black variety.

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[1] Hist. Nat. xxxiii. 8, 1.

[2] Hist. Nat. xxxiii. 6.

His description of *plumbum candidum*, and the state in which it was found, leaves no doubt that this much-valued metal was tin; it being represented as occurring among sand, in the dried-up beds of rivers, and as only known from the other substances with which it was found associated, by its dark colour and great weight. "There is likewise found in the gold mines a kind of lead ore which they call *elutia* (stream-tin). The water which is let into the mines washes, and carries down with it, certain little black stones, streaked and marked with white, and as heavy as the gold itself. It is gathered with that metal, and they remain together in the baskets in which the gold is collected." Again: "You cannot solder together two pieces of black lead without white lead, neither can this be united to the other without the aid of oil."

He also says of this metal: "Neither out of white lead can any silver be extracted; whereas out of the black this is commonly done."

When speaking of common lead, the same author says: "It is much used for conduit-pipes and for being hammered into thin plates;" and he then goes on to describe the mines of France, Spain, and Britain, which he states, when worked out and exhausted, become quite as productive as ever, and indeed even more so, if allowed to remain a short time without being worked; for which he accounts by supposing the metal to be produced by the air, which has then free access into the mine. With regard to the state in which *plumbum nigrum* occurs, we are informed that, "Black lead has a double origin; for it is either produced in a vein of its own, without any other metal; or otherwise it is mingled with silver in the same mine; being mixed together in the same stone of ore, and they are only separated by melting and refining in a furnace.[2] The first liquid that flows from the furnace is tin (*stannum*), and the second silver. That which remains behind is galena, the third element of the vein, which being again melted, after two parts of it are deducted, yields black lead."

The above passage is obscure: tin, lead, and silver are not often found in the same stone, but were they thus to occur, the tin would not be the first to flow out of the furnace. The ore from which the ancients produced their lead appears to have been galena, a name employed by Pliny as synonymous with molybdæna, which is described by Dioscorides and himself as an argentiferous lead ore.

Of the metals employed by the ancients for the manufacture of objects adapted to the everyday usages of life, copper and its alloys were the most common; as by far the greater portion of the coins, utensils, and implements of war, which are occasionally brought to light, are composed of some alloy of copper; and consequently, the making of these alloys, and their adaptation to the various wants of mankind, must have formed an important branch of the manufactures of the Greeks and Romans.

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[1] 'Ancient Mineralogy,' by N. F. Moore, LL.D., p. 60.

[2] Hist. Nat. xxxiv. 16.

Accordingly, the author of the "Natural History of the World," after describing the properties of this metal, and stating the localities in which that of the best quality was found, gives the composition and proportions employed in the various mixtures then common in Rome, and informs us to what uses they were severally applied. He also states that copper was first found in the island of Cyprus, whence two distinct kinds were exported.[1] One called *coronarium*, which, when reduced to thin leaves and coloured with the gall of an ox, had a golden colour, and was employed for making coronets and tinsel ornaments for actors, from which circumstance it derived its appellation. Another variety, which was named *regulare*, is not particularly described, except that, like the former, it would stand hammering, and might thus be made to take any required form.

The brass of next best quality came from Campania, where it was the custom to add eight pounds of lead to every hundred pounds of copper. It is also mentioned that in Gaul it was usual to melt copper among red-hot stones, for the purpose of obtaining a steady heat, as a quick fire was found to blacken the metal and render it brittle. He further informs us that the process was completed in one operation, but states that the quality would be improved by more frequent melting: "Moreover, it may not be amiss to state also that all kinds of brass melt best in the coldest weather. For statues and tables, brass is worked in the following manner: first the ore, or stone, as it comes out of the mine, is melted; and, as soon as this is done, they add to it a third part of scrap brass, consisting of broken pieces of vessels that have been used; for it is time and use alone that bring brass to perfection. It is the rubbing which conquers the natural harshness of the metal. They then mix twelve pounds and a half of tin to every hundred pounds weight of the aforesaid melted ore. The softest alloy is called *formall*, in which are incorporated a tenth part of black lead and one-twentieth part of argentine lead; it is this mixture which best takes the colour called *grecanic*. The last alloy is that which is called *ollaria*, or pot-brass, as it takes its name from the vessels for which it is mostly employed, and this is made by tempering every hundred pounds weight of brass with three or four pounds weight of argentine lead or tin."[2]

The alloys above described are merely modifications of bell-metal or bronze; but it is not improbable that the ancients were acquainted with zinc-brass long before this period. Aristotle tells us that the Mosynœcians, a people who inhabited a country not far from the Euxine Sea, were said to make copper of an exceedingly fine colour, not by the addition of tin, but by mixing and cementing it with an earth found in that country.[1] We are also informed by Strabo that, in the neighbourhood of Andêra, a city of Phrygia, a remarkable kind of stone was met with, which, being calcined, became iron, and, on being fluxed with a certain kind of earth, yielded drops of a silvery-looking metal, which, mixed with copper, formed an alloy called *orichalcum.*[2]

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[1] Hist. Nat. xxxiv. 8.

[2] Hist. Nat. xxxiv. 9.

Sextus Pompeius Festus, who abridged a work of Verrius Flaccus, a writer of the time of Augustus, mentions *cadmia*, which he describes as an earth thrown upon copper in order to convert it into orichalcum.[3]

On this subject Pliny affords us but little information, merely stating where cadmia was found, and naming some of its medicinal properties; but he seems to have regarded it rather as an earth which gave a yellow colour to copper, than as an ore of a distinct metal, zinc being in no instance mentioned by him, although he speaks of a kind of brass which was manufactured in the island of Cyprus from copper and cadmia.

If the foregoing quotations were not sufficient to show that the ancients were acquainted with zinc-brass, the fact is distinctly proved by the following analyses, published by the author in 1852:[4]—

| | 1. | 2. | 3. | 4. | 5. |
|---|---|---|---|---|---|
| Copper . . . | 82·26 | 81·07 | 83·04 | 85·67 | 79·14 |
| Zinc . . . | 17·31 | 17·81 | 15·84 | 10·85 | 6·27 |
| Tin . . . | ... | 1·05 | ... | 1·14 | 4·97 |
| Lead . . . | ... | ... | ... | 1·73 | 9·18 |
| Iron . . . | 0·35 | ... | 0·50 | 0·74 | 0·23 |
| | 99·92 | 99·93 | 99·38 | 100·13 | 99·79 |
| Specific gravity . | 8·52 | 8·59 | 8·50 | 8·30 | 8·83 |

No. 1—Large brass of the Cassius family, about B.C. 20; metal of a yellow colour. No. 2—Large brass of Nero, A.D. 60; reverse, Rome seated; metal, bright yellow. No. 3—Titus, A.D. 79; metal, yellow and soft. No. 4—Hadrian, A.D. 120; "Fortunæ reduci;" finely patinated; metal, fine yellow. No. 5—Faustina, A.D. 165; "Pietas," without patina; metal of a whitish colour and very brittle.

That metallic zinc, however, was known to the ancients, there is no evidence to show, since the metal mentioned by Strabo as given out in drops from a certain stone when heated, could scarcely have been zinc, which would have been volatilized if treated in the way described; and we may, therefore, suppose that, if the stone referred to by him was an ore of zinc, it might also have contained some other metal, such as lead, with which it is often associated, and which would produce the appearance in question. Ambrose, Bishop of Milan, describes the transformation of copper into orichalcum, as being effected by means of a drug, and not by the addition of another metal. From this we may infer that he was unacquainted with the metallic nature of the material employed, although, from his calling it a drug, he was perhaps aware of its possessing certain medicinal properties.

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[1] Arist. de Mirab., op. v.

[2] Strabo, Geo. lxiii.

[3] "Cadmia terra, quæ in æs conjicitur ut fiat orichalcum."—Fes. de Ver., Watson's 'Chemical Essays,' iv. p. 91.

[4] 'Quart. Journ. of Chem. Soc. of London,' 1852, iv. p. 252.

A similar description of the manufacture of brass is given by Primasius, Bishop of Adrumetum, in Africa, in the sixth century, and by Isodorus, Bishop of Seville, in the seventh. Agricola, who wrote in the sixteenth century, was also apparently ignorant that cadmia contained zinc, of which we have no authentic account until we find it mentioned by Paracelsus,[1] who died in 1541, from which it would appear that, although the manufacture of zinc-brass is of great antiquity, the extraction of the metal itself is a comparatively modern discovery.

Iron was doubtless employed in very early times, although the uses of copper and its alloys were probably known at a much earlier period. Hesiod speaks of iron as having been unknown during the age of bronze, and Lucretius says, with regard to this metal, "Et prior æris erat quam ferri cognitus usus."[2] Moses compares the deliverance of the Israelites from Egyptian bondage to their being "brought forth out of the iron furnace."[3] In the time of Homer iron was well known, but appears to have been employed more sparingly than bronze, and must have been of considerable value, since a mass of iron, which had been used by Eëtion as a quoit, is offered by Achilles as a prize at the funeral of Patroclus.

When the interpreter who accompanied Herodotus reads to him an inscription on one of the Egyptian pyramids relative to the amount of money expended on radishes, onions, and garlic, for the workmen employed on its construction, he makes the reflection, that if this were true, how much more must have been paid for iron tools, bread, and clothing.[4] If we allow that iron tools were used in building these monuments, this metal must have been in common use during some portion of the time which elapsed between the birth of Abraham and the captivity of Joseph.[5]

Aristotle says that iron is purified from scoria by melting, and when it had been treated thus several times and became pure, it was changed to steel, *στόμωμα*.

Daimachus, a writer contemporary with Alexander the Great, speaks of four different kinds of steel, and the purposes to which they were severally suited. "Of steels (*τῶν στομωμάτων*), there is the Chalybdic, the Synopic, the Lydian, and the Lacedæmonian. The Chalybdic is best for carpenters' tools, the Lacedæmonian for files, drills, gravers, and stone chisels; the Lydian also is suited for files, and for knives, razors, and rasps."[6]

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[1] "Marchasita Aurea," mentioned by Albertus Magnus in the 13th century, is believed by Beckmann and some others to have been zinc.

[2] Lib. v.

[3] Deut. iv. 20.

[4] Herod., ii. 125.

[5] Russell's Egypt, p. 89.

[6] See Stephanus De Urbibus, word Lacedæmon; and Fabricii Bib. Græc., vol. ii. p. 588 (Anc. Mineralogy, p. 59).

In speaking of iron, Pliny says: "After copper comes iron, both the most useful and most fatal instrument of life. With iron, man delves the earth, plants trees, prunes his orchards, trims his vines, cutting off the older branches, and thereby throwing more vigour into the grapes; by its aid man builds houses, cuts stone, and prepares a thousand other implements; but by it war, atrocity, and villainy are effected and rendered common."[1] He also describes iron as occurring in almost every part of the known world, but particularly in the island of Elba, where the colour of the earth indicates the presence of the ore.

Sulphide of antimony, called by the Greeks στίμμι, and by the Romans *stibium*, was from the earliest times, and is still, used in the East for tinging the eyebrows, &c.

Pliny's description of stibium as "candidæ nitensque"[2] does not suit in all respects common sulphide of antimony. In preparing it as a paint, it is, according to Dioscorides, to be enclosed in a lump of dough, and buried in coals until reduced to a cinder. After being extinguished with milk and wine, it is to be again placed upon coals and blown upon until ignition takes place, but if burned longer it becomes lead.[3]

Pliny directs cow-dung to be used in the place of dough, but varies so entirely from the recipe of Dioscorides, that it is evident he had some other authority before him. Yet he likewise recommends moderation in burning as especially necessary, lest it should be converted into lead (ne plumbum fiat).[4] The fair inference, therefore, is that the ancients occasionally saw antimony reduced to its metallic state, but failed to recognize it as a new metal.

## PHYSICAL PROPERTIES OF METALS.

The metals are a class of simple substances, possessed of a peculiar lustre, and having the property of conducting heat and electricity with facility. But both in their chemical and physical properties they differ much from one another, and are consequently applicable to a great variety of uses.

Colour.—Most of the metals, when in a finely-divided state, are of a grey colour, but, when consolidated and polished, approach more nearly to white. The colours of some of them are, however, very decided: thus copper is red, gold is yellow, and lead blue.

Alloys formed by the mixture of different metals usually possess to a certain extent the colours of the metals of which they are composed. Those resulting from the combination of two or more grey or white metals will themselves be grey or white. But, if a coloured metal enter into its composition, the alloy will assume its colour in a marked degree, although, if the proportion of the coloured metal be small compared with the amount of that which is not coloured, this is not always apparent. In some cases, however, as in that of the alloys of gold and silver, a comparatively small amount of a white metal has the effect of destroying the colour of the other.

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[1] Hist. Nat. xxxiii. 14.

[2] Hist. Nat. xxxiii. 33.

[3] Dioscor., v. 99.

[4] Hist. Nat. xxxiii. 34.

Opacity and Lustre.—The metals possess a great degree of opacity, and are remarkable for a peculiar lustre, known as metallic. All, however, are not equally opaque, as gold, when reduced to extremely thin leaves, transmits rays of green light. Silver leaf of one hundred-thousandth of an inch in thickness is opaque; but very thin leaves of an alloy of silver and gold appear of a blue colour when viewed by transmitted light. Iron in the thinnest electro films is transparent, as is also platinum.

The lustre of metals is a consequence of their great power of reflecting light. When reduced to the state of powder, their peculiar metallic appearance disappears, but is immediately reproduced by rubbing with a burnisher, or any other hard and smooth substance.

Hardness.—The metals differ from one another in no respect more than with regard to their hardness. Those which are pure are usually less hard than their alloys, and some of them are so soft as to admit of being easily scratched with the nail, or even moulded between the fingers.

The following table, arranged by Dumas, shows the relative degrees of hardness of some of the more common metals :—

| | | | |
|---|---|---|---|
| Titanium | Harder than steel. | Chromium | Scratch Glass. |
| Manganese | | Rhodium | |
| Platinum | Scratched by Calc Spar. | Nickel | Scratched by Glass. |
| Palladium | | Cobalt | |
| Copper | | Iron | |
| Gold | | Antimony | |
| Silver | | Zinc | |
| Tellurium | | Lead | Scratched by the nail. |
| Bismuth | | Potassium | Soft as wax at 15·5° C. |
| Cadmium | | Sodium | |
| Tin | | Mercury | Liquid at ordinary temperatures. |

Specific Gravity.—The specific gravities of the metals differ very widely, as among them we find some bodies more than twenty times heavier than water; whilst others weigh less than their bulk of that liquid.

The principal metals, arranged according to their specific gravities, are given in the following table. Water = 1 ; temp. = 15·5° C. (60° F.) :—

Table of Specific Gravities.[1]

| | | | |
|---|---|---|---|
| Platinum | 21·50 | Cobalt | 8·54 |
| Uranium | 18·40 | Manganese | 8·00 |
| Gold | 19·50 | Iron | 7·79 |
| Tungsten | 17·60 | Tin | 7·29 |
| Mercury | 13·59 | Zinc | 6·86—7·1 |
| Palladium | 11·30—11·80 | Antimony | 6·80 |
| Lead | 11·45 | Tellurium | 6·11 |
| Silver | 10·50 | Arsenic | 5·88 |
| Bismuth | 9·90 | Aluminium | 2·56—2·67 |
| Copper | 8·96 | Magnesium | 1·74 |
| Nickel | 8·50 | Sodium | 0·97 |
| Cadmium | 8·70 | Potassium | 0·86 |
| Molybdenum | 8·63 | Lithium | 0·59 |

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[1] Fownes, 'Elementary Chemistry,' 12th edit., p. 303.

Crystallization.—Most of the metals susceptible of direct use crystallize in the regular or cubic system, and among them gold, silver, and copper are commonly found in nature in cubes, octohedra, or other forms of this system. Tin is tetragonal or quadratic, and the closely allied elements, arsenic, antimony, bismuth, and tellurium are rhombohedral. Platinum is cubic, but some of its allied metals, such as osmium and iridium, are hexagonal.

In order to crystallize a metal artificially, it is often sufficient to melt a few ounces in a crucible, and, having permitted it to cool on the surface, to pierce the crust thus formed and allow the metal in the interior to flow out. By this means very beautiful crystals of bismuth may be obtained; but in the case of some of the less fusible metals larger masses and slower cooling are necessary to produce this effect, and consequently these are never found in a crystalline state unless considerable weights have been fused, and allowed gradually to cool, as sometimes occurs in the furnaces in which their metallurgical treatment is effected.

It also frequently happens that one metal may be precipitated in a crystalline form by placing a strip of another metal in a solution of its salts. In this way silver is deposited by mercury, and a piece of zinc placed in a solution of lead acetate precipitates the lead in feathery crystals. Gold is occasionally deposited in this form from ethereal solutions, and a stick of phosphorus produces the same effect. Nearly all the metals yield crystals when deposited from their solutions by electric currents of feeble intensity, and it is probably to this action that we are indebted for many of the beautiful specimens of the native metals which enrich the cabinets of mineralogists.

Malleability.—When a piece of metal is struck by a hammer, it either flattens under the blow or splits with more or less facility into fragments. To the former property the name of *malleability* is applied, whilst metals possessing the latter peculiarity are termed *brittle*. The malleable metals may be reduced into thin leaves either by the hammer or by the flatting-mill.

This consists of two metallic cylinders (A, B, fig. 1) placed horizontally one above the other. These, by means of cog-wheels, are made to revolve in contrary directions, as shown by the arrows. The rollers are so arranged in a frame as to admit of being placed, through the medium of strong screws, at any required distance from one another; or, if necessary, of being brought into actual contact. To reduce a piece of metal by this means to the form of a thin sheet, it should be first cast in the shape of a rectangular ingot, having nearly the same width as the required plate. One of its ends is then flattened into the shape of a wedge so as to enter easily between the rollers, which, on being set in motion, draw the metal in, and pass it through to the other side, reduced in thickness and proportionately elongated. By repeating this operation several times, and

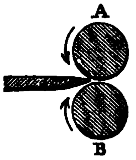

Fig. 1.

gradually reducing the distance between the two cylinders, sheets of almost any degree of thinness may be obtained.

During this compression of the metals, their molecular structure gradually undergoes a change, and those which at first are soft and pass readily through the mill, soon become brittle and difficult to work. They then require to be softened by being heated to redness, and afterwards cooled. This process is called *annealing*.

Gold is the most malleable of metals, and may be made into leaves of only $\frac{1}{200,000}$th of an inch in thickness, each grain of which will cover a surface of fifty-four square inches. The metals are arranged in the following list according to their malleability :[1]—

1. Gold.
2. Silver.
3. Copper.
4. Tin.
5. Platinum.
6. Lead.
7. Zinc.
8. Iron.
9. Nickel.

DUCTILITY.—The above-named metals are also ductile, or capable of being drawn into wire, but do not possess this property in the same order as their malleability. Wire is manufactured by passing an elongated piece of metal through the progressively diminishing holes of a steel tool, called a draw-plate. By this means wires of almost any diameter may be obtained, as the metal takes the size of the last hole through which it has passed. Silver, for the purposes of embroidery, is frequently made into wires $\frac{1}{500}$th of an inch in diameter. A grain of gold may be drawn into a wire 550 feet long by enveloping the ingot operated upon in a coating of silver, and then passing it through the draw-plate. The wire thus produced will also be found covered with silver, and on removing this latter metal by nitric acid, an enclosed gold wire, of only $\frac{1}{5,000}$th of an inch in diameter, will be obtained.

The following metals are arranged according to their ductility :—

1. Gold.
2. Silver.
3. Platinum.
4. Iron.
5. Nickel.
6. Copper.
7. Zinc.
8. Tin.
9. Lead.

TENACITY.—The power possessed by different metals of sustaining weights is very variable, and influences in a great degree the uses to which they can be applied. It is therefore important to ascertain by experiment their relative tenacities, and the various influences which affect them in this respect. For this purpose wires of equal lengths and diameters are sometimes employed. These are firmly suspended by one end, while to the other extremity weights are successively added until the rupture of the wire is effected. The weight which causes the wire to break necessarily represents the relative tenacity of the metal of which it is composed, when compared with others similarly treated.

The following table gives the approximate tensile strength, per square inch of section, of several of the more important metals :—

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[1] Regnault, 'Cours Élémentaire de Chimie,' i. 412.

| | Tons. | | Tons. |
|---|---|---|---|
| Antimony, cast | 0·47 | Iron, wrought, average | 22·00 |
| Bismuth „ | 1·45 | Lead, cast | 0·80 |
| Copper, bolts | 17·00 | „ sheet | 1·50 |
| „ cast | 8·40 | Silver | 18·20 |
| „ sheet | 13·40 | Steel | 52·00 |
| „ wire | 26·00 | „ plates | 35·00 |
| Gold | 9·10 | Tin, cast | 2·00 |
| Iron, cast, average | 7·30 | Zinc „ | 3·30 |

FUSIBILITY.—All the metals admit of being liquefied by the application of heat; but the temperatures at which they melt are extremely various. Mercury retains its liquid form during the most intense colds of our climate. Potassium and sodium fuse below the boiling-point of water. Tin melts at about 227° C.; lead at 325° C.; and antimony below redness. Gold, silver, and copper require a red heat; iron, nickel, and cobalt fuse at a white heat. Manganese and palladium are melted only by the strongest heat of a wind furnace; chromium, molybdenum, and tungsten agglutinate but slightly when treated in the same way; platinum, iridium, rhodium, titanium, &c., yield only to a powerful voltaic current, or to the flame of the oxyhydrogen blowpipe.

TABLE OF FUSIBILITY OF METALS.[1]

| | | Melting Points. | |
|---|---|---|---|
| | | F. | C. |
| Fusible below a red heat. | Mercury | −39° | −39·44° |
| | Rubidium | +101·3 | +38·5 |
| | Potassium | 144·5 | 62·5 |
| | Sodium | 207·7 | 97·6 |
| | Lithium | 356 | 180 |
| | Tin | 442 | 227·8 |
| | Cadmium | about 442 | 228 |
| | Bismuth | 497 | 258 |
| | Thallium | 561 | 294 |
| | Lead | 617 | 325 |
| | Tellurium | Rather less fusible than lead. | |
| | Arsenic | Unknown. | |
| | Zinc | 773 | 412 |
| | Antimony | Just below redness. | |
| Infusible below a red heat. | Silver | 1,873 | 1,023 |
| | Copper | 1,996 | 1,091 |
| | Gold | 2,016 | 1,102 |
| | Cast-iron | 2,786 | 1,530 |
| | Pure iron | Highest heat of forge. | |
| | Nickel | | |
| | Cobalt | | |
| | Manganese | | |
| | Palladium | | |
| | Molybdenum | Agglomerate, but do not melt in forge. | |
| | Uranium | | |
| | Tungsten | | |
| | Chromium | | |
| | Titanium | Infusible in ordinary blast-furnaces; fusible by oxyhydrogen blowpipe. | |
| | Cerium | | |
| | Osmium | | |
| | Iridium | | |
| | Rhodium | | |
| | Platinum | | |
| | Tantalum | | |

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[1] Fownes, 'Manual of Elementary Chemistry,' 12th ed., p. 304.

POWER OF CONDUCTING HEAT, &c.—Some of the metals transmit heat with much greater facility than others. In the following table the metals are arranged in the order of their decreasing conducting powers, and opposite to the name of each body is placed the approximate ratio of the facility with which it transmits heat:—

RELATIVE CONDUCTIVITY OF METALS, &c.—SILVER = 100.[1]

| | At 12° C. | | At 12° C. |
|---|---|---|---|
| Silver | 100·0 | Steel | 11·6 |
| Copper | 73·6 | Lead | 8·5 |
| Gold | 53·2 | Platinum | 8·4 |
| Brass | 23·6 | German silver | 6·3 |
| Tin | 14·5 | Rose's fusible metal | 2·8 |
| Iron | 11·9 | Bismuth | 1·8 |

The conductivity of the various metals for electricity is approximately in the same ratio as their capability of transmitting heat.

SPECIFIC HEAT.—The amount of heat required to raise equal weights of different metals from the same to another given temperature is very variable. Thus, if we express by 1 the quantity necessary to raise a weight of water from 0° C. to 1° C., that which must be supplied in order to elevate the same weight of the following metals to the same temperature will be as below:—

| | | | |
|---|---|---|---|
| Iron | 0·1138 | Cadmium | 0·0567 |
| Nickel | 0·1086 | Tin | 0 0562 |
| Cobalt | 0·1070 | Antimony | 0·0508 |
| Zinc | 0·0956 | Platinum | 0·0311 |
| Copper | 0·0951 | Gold | 0·0324 |
| Palladium | 0·0593 | Lead | 0 0314 |
| Silver | 0·0570 | Bismuth | 0·0308 |

DILATATION BY HEAT.—Metals crystallizing in the cubic system are expanded by heat similarly in all directions, without regard to their structure; but those occurring in rhombohedral forms behave differently in this respect, the elongation for a similar increment of heat being much more rapid parallel to the principal crystallographic axis than perpendicular thereto.

The following table of the computed elongation of metals from 0° to 100° C., is due to the researches of Fizeau:—

| | | | |
|---|---|---|---|
| Gold, melted | 0·001451 | Aluminium | 0·002336 |
| Silver " | 0·001936 | Tin (compressed powder) | 0·002269 |
| Platinum " | 0·000916 | Zinc " " | 0·002905 |
| Platiniridium (8% iridium) | 0·000890 | Cadmium " " | 0·003102 |
| Palladium, annealed | 0·001189 | | |
| Copper, native | 0·001708 | Bismuth, parallel to principal axis | 0·001642 |
| " melted | 0·001698 | | |
| Iron, soft malleable | 0·001228 | Bismuth, perpendicular to principal axis | 0·001239 |
| Cast-steel, hard | 0·001362 | | |
| " " soft | 0·001113 | Antimony, parallel to principal axis | 0·001683 |
| Cast-iron, grey | 0·001075 | | |
| Lead, melted | 0·002948 | Antimony, perpendicular to principal axis | 0·000895 |

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[1] Poggendorff's Ann. b. 89, p. 497.

VOLATILITY.—All metals are more or less volatile, although a certain number only admit of being readily converted into vapour even at the highest temperatures of our furnaces. Among the more volatile metals are—zinc, cadmium, mercury, arsenic, tellurium, potassium, and sodium. Several others have the property of communicating characteristic colours to flame, and are therefore evidently to a small extent volatile.

ALLOYS.—The metals are generally capable of uniting with one another, and forming a class of compounds possessing more or less the properties of their several constituents. Alloys are usually more fusible and harder than the metals which enter into their composition; and as these properties may be regulated according to the relative amounts of the various metals employed, an infinite number of modifications may be obtained. Thus copper is malleable and ductile, but is somewhat difficult to fuse, and for many purposes does not possess the requisite hardness. In many instances these defects may be obviated by the addition of zinc, which, without much impairing its malleability, renders it fusible, heightens its colour, and at the same time communicates to it a proper degree of hardness.

By the addition of ten parts of tin to ninety of copper, an alloy known as gun-metal is obtained, which is also used, under the name of bronze, for bearings, statues, and various ornamental purposes.

For printers' type an alloy is required at the same time hard and fusible, and which does not materially contract in cooling. Lead, which is a fusible metal, is evidently unfitted for this purpose by its softness, whilst antimony and bismuth are too liable to break under the pressure to which type is exposed in the operation of printing. By combining, however, antimony and lead, with sometimes a little zinc or tin, an alloy is produced which fulfils all these conditions, and furnishes a material well adapted for the purpose intended.

It has been stated that, by alloying the metals, we obtain compounds possessed of very different ductility, malleability, hardness, and colour, from those belonging to the bodies which enter into their composition. Thus gold and lead, and gold and zinc, form brittle alloys, and a minute quantity of arsenic added to copper renders it white and brittle. It is also to be observed that an alloy composed of two metals has seldom a density corresponding to the mean obtained, by calculation, from the relative amounts and specific gravities of its constituents.

The following table from Thénard (Traité de Chimie, vol. i. p. 394) shows in what cases the specific gravity of the compound is superior, and when inferior to the mean of that of the combined metals. It is, however, doubtful whether some of the mixtures specified should be regarded as alloys:—

| Alloys possessed of greater specific gravity than the mean of their components. | Alloys having a specific gravity inferior to the mean of their components. |
|---|---|
| Gold and Zinc. | Gold and Silver. |
| „ Tin. | „ Iron. |
| „ Bismuth. | „ Lead. |
| „ Antimony. | „ Copper. |
| „ Cobalt. | „ Iridium. |
| Silver and Zinc. | „ Nickel. |
| „ Lead. | Silver and Copper. |
| „ Tin. | Copper and Lead. |
| „ Bismuth. | Iron and Bismuth. |
| „ Antimony. | „ Antimony. |
| Copper and Zinc. | „ Lead. |
| „ Tin. | Tin and Lead. |
| „ Palladium. | „ Palladium. |
| „ Bismuth. | „ Antimony. |
| „ Antimony. | Nickel and Arsenic. |
| Lead and Bismuth. | Zinc and Antimony. |
| „ Antimony. | |
| Platinum and Molybdenum. | |
| Palladium and Bismuth. | |

The action of acids on alloys varies according to the relative amounts of their constituents. Silver alloyed with a large quantity of gold is protected from the action of nitric acid, by which, under ordinary circumstances, it is readily attacked. Sometimes, however, the reverse of this takes place, and metals which are totally insoluble in certain menstrua are made to dissolve in them by the addition of a metal on which they have the power of acting. In this way, platinum, although of itself insoluble in nitric acid, may be dissolved by it when sufficiently alloyed with silver. Alloys consisting of two metals, the one easily oxidizable, the other possessing a less affinity for oxygen, may be readily decomposed by the combined action of heat and air. In this case the former metal will be rapidly converted into an oxide, excepting perhaps the last portions, which may in some degree be protected from further action by the oxide already formed. The increased affinity for oxygen exhibited by the more oxidizable metal, in presence of another less affected by this agent, is probably an electrical phenomenon; the action being in some cases so rapid as to produce combustion. This occurs when an alloy of three parts of lead and one of tin is strongly heated in contact with air.

As the chemical properties of metals belong rather to the science of chemistry than to metallurgy, and as the limits of the present work do not admit of this subject being comprehensively treated, it has been thought proper, in this place, to omit it altogether. The study of metallurgy can, however, only be profitably undertaken by those possessing a competent knowledge of chemistry, which must be acquired by a careful study of good text-books, supplemented by a considerable amount of laboratory experience.

## FUEL.

Any substance which admits of being rapidly oxidized or burned by atmospheric air, and evolves during that operation an amount of heat capable of being applied to economic purposes, is called a fuel. The elements that are so applied in metallurgy are carbon, hydrogen, sulphur, silicon, and phosphorus; but as the applications of the three latter are only special, the term "fuel" is generally restricted to carbon and hydrogen, and their compounds; the natural gaseous, liquid and solid hydrocarbons; wood, peat, and coal, with the artificial products, charcoal, coke, and sundry gaseous products susceptible of oxidation (furnace and producer gases); the fuel value of the latter being essentially due to carbonic oxide or carbon monoxide. In all fuels containing carbon, hydrogen, and oxygen, the proportion of hydrogen may be equal to or greater than that required to form water with the oxygen, but is never less. In such combinations only the hydrogen in excess is available as a source of heat; so that in the combustion of a substance of which the composition may be regarded as carbon and water, the carbon alone is the source of heat. Indeed, in such cases the hydrogen is the cause of the loss of a considerable amount of otherwise available heat, since it may be viewed as existing in combination with oxygen in the state of water, which must be evaporated at the expense of a portion of the heat developed by the combustion of carbon.

The products of the perfect oxidation or complete combustion of carbon and hydrogen are respectively carbon dioxide or carbonic acid (now commonly called carbonic anhydride) and water; and these products are likewise obtained on the combustion of any hydrocarbon or carbohydrate. The amount of heat developed by the complete combustion of any elementary substance in the same allotropic condition is perfectly definite, and is the same whether the combustion be effected rapidly or otherwise. By the perfect or complete combustion of carbon is understood its conversion into carbonic anhydride; when applied to hydrogen these terms imply the degree of oxidation necessary to produce water. In the case of carbon perfect combustion results in the formation of its highest, and, at the same time, most stable oxide. With respect to hydrogen it is somewhat different; water is the most stable oxide of hydrogen, but it is not its highest oxide. Peroxide of hydrogen contains twice the amount of oxygen that water does, but the affinity by which the second atom of that element is retained is exceedingly feeble.

The pyrometric degree, or *intensity*, of heat is perfectly distinct from and independent of the *quantity* of heat developed by combustion. The quantity of heat generated on the perfect combustion of a given weight of one body may be much greater than that produced by the complete combustion of a similar weight of another body, but the intensity of the heat in the second case may far exceed that obtained in the first. All other circumstances being the same, the intensity of the heat developed by the combustion of a given body will be directly proportionate to the rapidity of the operation or inversely as the time occupied in effecting it. The term *calorific intensity* is employed in contradistinction to *calorific power*, which expresses the quantity of heat evolved by combustion.

CALORIFIC POWER OF FUELS.—Various methods have been employed for the purpose of measuring the *relative* amounts of heat evolved by the combustion of equal weights of different bodies; and as this heat cannot be directly estimated, it must, in all cases, be determined in accordance with certain effects produced. Any effect of heat may be employed as a means of measuring its quantity by applying the principle that when two equal portions of the same substance, in the same state, are acted on by heat in the same way, so as to produce the same effect, the quantities of heat are equal. It is first necessary to choose a standard of comparison, and to determine the effect of heat upon that body. We may thus choose a given weight of ice at the freezing-point as the standard, and we may define as the unit the quantity of heat which must be applied to this weight of ice to convert it into water, still at the freezing-point. By this system the quantity of heat is measured by the number of pounds, grammes, or other agreed weight of ice, at the freezing-point, which that quantity of heat would convert into water at the freezing-point. We may also employ a different system of measurement by defining a quantity of heat as measured by the weight of water at the boiling-point, which it would convert into steam of the same temperature. This method is often employed in practically determining the calorific value of fuels.

Another method is to define as the unit that quantity of heat which, if applied to a unit mass of water, at some standard temperature, will raise that water one degree Centigrade. According to this method a quantity of heat is measured by the amount of water, at a standard temperature, which that amount of heat would raise one degree. The amount of heat required to raise one gramme of water from 0° to 1° C. is now generally adopted as the *unit of heat* or *calorie*.[1]

All that is assumed in these methods of measurement is, that if it takes a certain quantity of heat to produce a given effect on a defined weight of water, to produce the same effect on another similar weight of water will require an equal amount of heat; or that twice the quantity of heat will be required for twice the quantity of water, and so on. It has been found that more heat is required to raise a given weight of water near the boiling-point one degree than at lower temperatures. If heat be measured according to either of the methods described, quantities of heat may be treated as mathematical values, and may be added, subtracted, &c., as required.

In the experiments of Rumford the apparatus employed consisted of a rectangular vessel of thin sheet-copper, enclosing a worm of three horizontal coils. This vessel was eight inches long, four and a half inches broad, and four and a half inches deep; containing the worm made of flattened copper tube, one inch in breadth and half-an-inch in depth. The lower end of the worm was secured to a circular hole in the bottom, near one of the square ends, while the other extremity issued vertically from the top of the vessel near its opposite extremity. A funnel-shaped copper mouthpiece, one and a half inch diameter, was fitted to the lower opening of the worm, and a tube inserted in the top of the box received a thermometer, by means of which the mean temperature of the water it contained could be determined. The substance of which it was sought to ascertain the calorific value was burned beneath and within the funnel-shaped mouth of the coil. The resulting current of air, after communicating the heat developed to the worm, and thence to the surrounding water, finally escaped from the other extremity. In order to avoid loss of heat by radiation, the temperature of the water with which the vessel was filled was reduced, immediately before the commencement of the experiment, a few degrees below that of the surrounding atmosphere, and the combustion was continued until the temperature of the water was raised exactly the same number of degrees above that of the air. It was estimated that in this way the vessel would receive as much heat by conduction and radiation as it lost during the experiment. In order to reduce as much as possible the loss of heat by conduction the apparatus was supported on slender wooden pillars. To test the capability of the instrument to absorb the whole of the heat developed by the combustion of the substance under examination, the resulting gaseous products were conducted through a second apparatus similar to the first, and were found not to augment the temperature of the water which it contained.

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[1] The British heat-unit is the amount necessary to raise 1 lb. of water 1° Fahr.

The data required in order to determine the calorific value of a substance by the use of this calorimeter are as follow:—

| | |
|---|---|
| Weight of substance consumed . . . . . | $n$ |
| " water . . . . . . . . | $w$ |
| " copper . . . . . . . . | $c$ |
| Specific heat of copper . . . . . | $s$ |
| Initial temperature of water . . . . | $t$ |
| Final temperature of water . . . . | $t'$ |

In experiments of great precision it would be necessary to make corrections for the glass of the thermometer, &c.; but in the determinations of Rumford such extreme accuracy was not attempted. In order to ascertain the weight of water which, in respect to absorption of heat, would be equivalent to the copper of which the apparatus is made, its weight must be multiplied by the specific heat of copper.

Let $x$ represent the amount of heat produced by the combustion of one part of any given body in atmospheric air; the following formula will then express the calorific value of that body:—

$$nx = (t' - t) \times (w + cs)$$

$$\therefore \quad x = \frac{(t' - t) \times (w + cs)}{n}$$

For example :—

Let $n$ = 20 parts by weight.
$w$ = 17,500 „
$c$ = 2,000 „
$s$ = 0·09515
$t$ = 4° C.
$t'$ = 13° C.

Then

$$x = \frac{(13 - 4) \times [17,500 + (2,000 \times 0\cdot09515)]}{20} = 7,960;$$

is the calorific value of the body under examination. Lavoisier, Dulong, Despretz, and Grassi have investigated the calorific values of various bodies, but more recent researches on this subject have been made by Favre and Silbermann and by Andrews. The apparatus employed by the later observers was, in principle, similar to that of Rumford, but was so constructed as to afford more accurate results; and all necessary corrections were made in the various calculations.

*Calorific Power of Carbon.*—The more recent experiments on carbon, in the different states of diamond, graphite, and charcoal, have afforded results agreeing very closely with each other, and the discrepancy in the results of the earlier observers may be accounted for by their having been ignorant that the combustion of carbon, even in oxygen, always gives rise to the production of a certain amount of carbonic oxide.(1) The incomplete oxidation of carbon, resulting in the production of carbonic oxide, is attended with the evolution of much less heat than is produced by its complete oxidation and conversion into carbonic anhydride. As, during the combustion of carbon, a certain amount of carbonic oxide is invariably produced, it becomes necessary to determine with accuracy the quantity of this gas resulting from each experiment. In order to do this, the products of combustion are first passed through a solution of potash, by which the $CO_2$ is absorbed, and subsequently through a tube containing cupric oxide heated to redness. In this way the CO is converted into $CO_2$, which is passed through another solution of potash and weighed. By these means not only the relation between the quantities of CO and $CO_2$ may be determined, but the total amount of carbon consumed is also ascertained. From the mean results of a considerable number of experiments 8,080 has been decided on as the calorific power of carbon existing in the form of purified wood charcoal.

*Calorific Power of Carbon Monoxide.*—In order to ensure the perfect combustion of carbonic oxide, or carbon monoxide, in the calorimeter, Favre and Silbermann found it necessary to mix it with one-third of its volume of hydrogen. To ascertain in each experiment the relative proportion of the two gases, a portion of the mixture was passed over heated cupric oxide, and the $CO_2$ and $H_2O$ produced estimated in the usual way. The mean of two experiments, after correction for hydrogen, gave 2,403 units of heat as the result of converting 1 gramme of CO into $CO_2$.

It follows that the amount of CO containing 1 gramme of carbon will evolve 5,607 units of heat by conversion into $CO_2$. One gramme of carbon is, however, found to evolve 8,080 units of heat in passing to the state of $CO_2$. Therefore one gramme of carbon by conversion into CO, will evolve $(8{,}080 - 5{,}607) = 2{,}473$ units. It is remarkable that carbon in passing to the state of CO should evolve less than one-third the heat developed by its conversion into $CO_2$. This may probably be accounted for by the large amount of heat rendered latent on the passage of carbon from the solid to the gaseous state, in combining with the first atom of oxygen.

*Calorific Power of Hydrogen.*—From the mean of six determinations Favre and Silbermann deduced 34,462 as the calorific power of hydrogen. In such experiments the weight of hydrogen consumed is calculated from that of the water collected.

*Berthier's Process for Estimating the Calorific Power of Fuels.*—On the assumption of the correctness of Welter's theory, that the heat evolved by combustion is directly proportionate to the amount of oxygen consumed, Berthier proposed to estimate the calorific power of fuels by burning them into $CO_2$ by means of the oxygen contained in litharge, and to ascertain the amount of oxygen abstracted, from the weight of the buttons of lead produced. In the case of pure carbon, or fuel consisting of carbon without any mixture of other reducing agents, this process may be employed with advantage, and is capable of affording approximately accurate results. But when disposable hydrogen is present, which is nearly always the case, even in coke and charcoal, the results obtained are no longer so satisfactory. This will be evident from the following considerations. One part by weight of hydrogen will reduce the same amount of monoxide of lead as three parts by weight of carbon. The calorific powers of hydrogen and carbon are, however, respectively 34,462 and 8,080, or, in round numbers, as 34 : 8. It consequently follows that the calorific power of 1 of hydrogen as compared with that of 3 of carbon is as 34 : 24; so that the weight of lead which would in the case of hydrogen represent a calorific power of 34, would in that of carbon be represented by 24. It is, therefore, evident that this process is inapplicable to the exact determination of the calorific powers of fuels containing variable quantities of carbon and hydrogen. As, however, hydrogen in excess of the quantity required to form water will alone have any reducing influence on oxide of lead, the amount of error is less than might be anticipated. Calculations based on Welter's theory, made on results obtained by Berthier's method, cannot afford absolutely correct results; but they deviate so little from the truth that this method, owing to its simplicity, is still in use, and is of considerable practical value. The results are never more than one-ninth smaller than those found by calculation on an analysis of the fuel, and the richer in carbon the substance is, and the less CO that has, through a careful execution of the assay, been formed during the experiment, the more nearly will they approximate to the truth.

Berthier's process consists in heating a known weight of the substance in fine powder, intimately mixed with a large excess of litharge, which, being de-oxidized by the combustible constituents of the fuel, yields a weight of lead proportionate to the quantity of those substances present. Every atom of oxygen abstracted from litharge will necessarily give rise to the reduction of an atom of metallic lead, and, consequently, a tolerably accurate measure of the relative heating values of different kinds of fuel is obtained by weighing the buttons of lead, produced under perfectly similar conditions.

TABLE OF CALORIFIC POWERS.

| One Gramme of each Substance burnt. | Supporter of Combustion. | Product of Combustion. | Number of Grammes of Water heated 1° C. | Observers. |
|---|---|---|---|---|
| Diamond . . . . | Oxygen | $CO_2$ | 7,770 | Favre and Silbermann. |
| Graphite, native . . | „ | „ | 7,811·5 | „ „ |
| „ artificial . . | „ | „ | 7,787·5 | „ „ |
| Carbon from gas retort . | „ | „ | 8,047·3 | „ „ |
| Charcoal from wood . . | „ | „ | 8,080 | „ „ |
| „ „ . . | „ | „ | 7,900 | Andrews. |
| „ „ . . | „ | CO | 2,227 | „ |
| „ „ . . | „ | „ | 2,473 | Favre and Silbermann. |
| Carbonic oxide . . . | „ | $CO_2$ | 2,402·7 | „ „ |
| Hydrogen gas . . . | „ | $H_2O$ | 34,462 | „ „ |
| „ „ . . . | „ | „ | 33,808 | Andrews. |
| Sulphur, native . . . | „ | $SO_2$ | 2,220·9 | Favre and Silbermann. |
| „ sublimed . . | „ | „ | 2,307 | Andrews. |
| Phosphorus . . . | „ | $P_2O_5$ | 5,747 | „ |
| Zinc . . . . . | „ | ZnO | 1,301 | „ |
| Iron . . . . . | „ | $Fe_3O_4$ | 4,134 | „ |
| Silicon . . . . | „ | $SiO_2$ | 7,830 | Troost & Hautefeuille. |

The following table gives Rumford's results as obtained by his water calorimeter:—

| One pound of the following kinds of Wood, when burnt, will heat | Pounds of Water from 0° to 100° C. |
|---|---|
| 1. Limetree wood. | |
| Dry wood, 4 years old . . . . . . . . | 34·707 |
| „ „ slightly dried . . . . . . . | 38·833 |
| „ „ strongly dried . . . . . . . | 40·131 |
| 2. Beech wood. | |
| Dry wood, 4 or 5 years old . . . . . . . | 33·798 |
| „ „ strongly dried . . . . . . . | 36·476 |
| 3. Elm wood. | |
| Wood dried, 4 or 5 years old . . . . . . | 30·205 |
| „ strongly dried . . . . . . . | 34·083 |
| „ dried brown . . . . . . . . . | 30·900 |
| 4. Oak wood. | |
| Common firewood, in small shavings . . . . . | 26·272 |
| The same, in thicker shavings . . . . . . | 25·590 |
| „ in thicker shavings . . . . . . | 24·748 |
| „ dried in the air . . . . . . | 29·210 |
| Very dry wood, in thin shavings . . . . . | 29·838 |
| „ „ thicker „ . . . . . . . | 26·227 |

| One pound of the following kinds of Wood, when burnt, will heat | Pounds of Water from 0° to 100° C. |
|---|---|
| 5. Ash wood. | |
| Common dry wood | 30·666 |
| The same, dried in air, shavings | 33·720 |
| „ shavings dried in an oven | 35·449 |
| 6. Sycamore wood. | |
| Strongly dried in an oven | 36·117 |
| 7. Wood of Mountain Ash. | |
| Strongly dried in an oven | 36·130 |
| Dried brown | 32·337 |
| 8. Wood of Bird Cherry. | |
| Dried wood | 33·339 |
| Strongly dried in an oven | 36·904 |
| Dried brown | 34·736 |
| 9. Fir wood (Deal). | |
| Ordinary dry wood | 30·332 |
| Well dried in the air, shavings | 34·000 |
| „ „ in an oven, shavings | 37·379 |
| „ „ brown, in shavings | 33·358 |
| „ „ in thick shavings | 28·695 |
| 10. Poplar wood. | |
| Wood dried in the ordinary manner | 34·601 |
| „ strongly dried in an oven | 37·161 |
| 11. Hornbeam. | |
| Dried wood (ordinary) | 31·704 |

The table on next page, from the Report on Coals suited to the Steam Navy, by De la Beche and Playfair, shows the effects produced by several varieties of coal, when burnt, in a well-constructed steam boiler, as well as those theoretically possible; together with the relation existing between the calorific values of their several constituents.

Calorific Intensity of Fuel.—By the pyrometric heating power, or calorific intensity, of a fuel, is understood the degree of temperature which may be obtained by its complete combustion in atmospheric air. This depends not only on its composition, but on various other circumstances, such as the purity, dryness, and temperature of the air employed as a supporter of combustion; the extent of the area of contact which the fuel offers to the air in a unit of time; the greater or less pressure under which the combustion takes place; the nature of the products; and the circumstances under which they are formed, &c., &c. Loss of heat from conduction and radiation likewise exerts a considerable influence on the practical results. It is, therefore, evident that to be enabled to calculate the calorific intensity of a fuel from its ultimate composition and the calorific power of its constituents, it will not only be necessary to take into consideration the nature of the resulting products, but also to introduce a number of factors, which complicate the formulæ, and which must, without care, become sources of error.

Various methods have been devised for the measurement of very elevated temperatures, but the indications of the earlier pyrometers having been found inaccurate, they have generally fallen into disuse. A method of measuring high temperatures, which may sometimes be found convenient, has been invented by Byström. It consists in exposing a ball of platinum of a given weight to the action of the heat to be measured, and then rapidly transferring it, through an inclined clay tube, to a vessel containing an ascertained weight of water of known temperature. From the increase of temperature experienced by the water is calculated the calorific intensity to be measured. Thus, if a ball of platinum weigh 100 grammes, and the water 200 grammes, the temperature being 15·5° C., and the heated platinum, when dropped into the water, raises its temperature to 32·2° C., then 32·2° – 15·5° = 16·7°, which multiplied by 2 (the weight of water being twice that of platinum) gives 33·4° as the temperature to which a weight of water equal to that of the platinum would have been elevated. In order to ascertain the temperature to which the platinum has been exposed, this must be multiplied by 32·1, the specific heat of water as compared

| Locality. | Number of lbs. of Water convertible into Steam from 100° C., by the Coke left by Coal. — Theoretical. | Number of lbs. of Water convertible into Steam from 100° C., by the carbon of Coal. — Theoretical. | Number of lbs. of Water convertible into Steam from 100° C., by the hydrogen of Coal. — Theoretical. | Total number of lbs. of Water convertible into Steam from 100° C., by 1 lb. of Coal. — Theoretical. | Actual number of lbs. of Water converted into Steam from 100° C., by 1 lb. of Coal. — Practical. | Percentage of Coke left by each Coal. |
|---|---|---|---|---|---|---|
| *Welsh Coals.* | | | | | | |
| Graigola . . . | 11·301 | 11·660 | 1·903 | 13·563 | 9·35 | 85·5 |
| Anthracite (Jones and Aubrey) . . . | 12·554 | 12·563 | 2·030 | 14·593 | 9·46 | 92·9 |
| Old Castle Firey Vein | 10·601 | 12·046 | 2·890 | 14·936 | 8·94 | 79·8 |
| Ward's Firey Vein . | ,, | 12·072 | 2·542 | 14·614 | 9·40 | ,, |
| Binea . . . | 11·560 | 12·181 | 2·912 | 15·093 | 9·94 | 88·10 |
| Llangennech . . | 10·599 | 10·741 | 2·519 | 14·260 | 8·86 | 83·69 |
| Pentrefelin . . | 10·841 | 11·749 | 2·038 | 13·787 | 6·36 | 85·0 |
| Powell's Duffryn . | 11·134 | 12·126 | 2·966 | 15·092 | 10·15 | 84·3 |
| Mynydd Newydd . | 9·831 | 11·463 | 3·441 | 14·904 | 9·52 | 74·8 |
| Three-Quarter Rock Vein . . . | 7·081 | 10·325 | 2·781 | 13·106 | 8·84 | 62·5 |
| Cwm Frood Rock Vein | 8·628 | 11·300 | 3·488 | 14·788 | 8·70 | 68·8 |
| Cwm Nanty Gros . | 8·243 | 10·767 | 3·165 | 13·932 | 8·42 | 65·6 |
| Resolven . . . | 10·234 | 10·899 | 3·072 | 13·971 | 9·53 | 83·9 |
| Pontypool . . . | 8·144 | 11·088 | 3·207 | 14·295 | 7·47 | 64·8 |
| Bedwas . . . | 8·897 | 11·075 | 3·766 | 14·841 | 9·79 | 71·7 |
| Ebbw Vale . . | 10·441 | 12·335 | 3·300 | 15·635 | 10·21 | 77·5 |
| Porthmawr, Rock Vein | 6·647 | 10·263 | 2·548 | 12·811 | 7·58 | 63·1 |
| Coleshill . . . | 6·468 | 10·145 | 2·654 | 12·799 | 8·00 | 56·0 |
| *Scotch Coals.* | | | | | | |
| Dalkeith Jewel Seam | 6·239 | 10·242 | 2·071 | 12·313 | 7·08 | 49·8 |
| Dalkeith Coronation Seam . . . | 6·924 | 10·570 | 2·202 | 12·772 | 7·71 | 53·5 |
| Elgin Wallsend . . | 6·560 | 10·454 | 2·968 | 13·422 | 8·46 | 58·45 |
| Fordel Splint . . | 6·560 | 10·933 | 2·884 | 13·817 | 7·56 | 52·03 |
| Grangemouth . . | 7·292 | 10·970 | 2·722 | 13·692 | 7·40 | 56·6 |
| *English.* | | | | | | |
| Broomhill . . . | 7·711 | 11·225 | 3·638 | 14·863 | 7·30 | 59·2 |
| *Irish.* | | | | | | |
| Slievardagh (Anthracite) . . . | 10·895 | 10·995 | 1·487 | 12·482 | 9·85 | 90·1 |

with platinum; that of the latter being represented by 1. Therefore, 33·4° × 32·1° = 1,072·14° C., which will be the temperature required. When, however, it becomes necessary to determine the temperature of the interior of a furnace, it is impossible to transfer the substance heated to the water in which it has to be cooled without the loss of a certain amount of heat; and the result obtained must consequently be to some extent incorrect. The electric pyrometer of Siemens, which depends in its action on the influence of heat on the conductivity of a platinum wire, and that employed by Schinz, which is a modification of the thermo-electric pyrometer of Becquerel, are said to afford satisfactory results. A description of these instruments would, however, occupy more space than the limits of the present work will allow, and, consequently, those who feel specially interested in the subject of the temperatures of blast-furnaces, and in the chemical and other conditions influencing the combustion of fuels therein, will do well to consult the works of Schinz and Bell, which are valuable additions to the literature of this branch of metallurgical investigation.[1]

When calcium carbonate is heated in a closed vessel from which the air has been carefully exhausted, an atmosphere of carbonic anhydride is produced by the dissociation of the constituents of the mineral; and it has been stated that the density of this atmosphere varies with its temperature in accordance with the law regulating the tension of aqueous vapour at low temperatures.

Although the accuracy of this statement has been denied, Lamy has made use of this principle in the construction of his pyrometer, which consists of a closed vessel exhausted of air, and containing calcium carbonate. To this vessel a manometer is attached, so that the pressure of the carbonic anhydride within may be read off and the temperature calculated therefrom in accordance with the above law. In order to be enabled to make the required calculation it is necessary that the pressure exerted by the carbonic anhydride at two or three known temperatures should have been previously ascertained.

The terms ordinarily employed to indicate high temperatures, such as red heat, white heat, &c., are very indefinite, since in judging of temperature by the eye, much must depend on the observer, and on the conditions with regard to light under which the observations may be made.

Pouillet, who examined high degrees of temperature by means of an air thermometer provided with a platinum bulb, arrived at the following results:—

| | | | |
|---|---|---|---|
| Incipient red corresponds to . | 525° C. | Deep orange . . | 1,100° C. |
| Dull red . . . . . . | 700° | Clear orange . . | 1,200° |
| Incipient cherry-red . . | 800° | White . . . . | 1,300° |
| Cherry-red . . . . | 900° | Bright white . . | 1,400° |
| Clear cherry-red . . . | 1,000° | Dazzling white . | 1,500° to 1,600° |

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[1] 'Researches on the Action of the Blast-Furnace,' by C. Schinz; translated by William H. Maw and Moritz Müller. Spon. 'Chemical Phenomena of Iron-Smelting,' by I. Lowthian Bell. Spon.

## ANALYSIS OF FUELS, &C.

Fuels which possess the highest calorific powers are not in all cases to be selected for practical purposes, as they may be subject to disadvantages which more than counterbalance this property. It is, therefore, necessary to ascertain by experiment what are the peculiar characteristics of each, so as to be enabled to select from among a number, such as may be most economically employed for the particular purposes to which they are to be applied.

ESTIMATION OF ASH.—If the substance to be examined be a wood, it should be first reduced to fine powder by means of a rasp, or if it be friable, such as charcoal, pit-coal, or coke, it may be pounded in an iron mortar.

A weighed portion (about 1 gramme) of the finely divided fuel, after having been passed through a sieve, is placed either in a platinum or porcelain crucible, and ignited over a gas-flame, until the whole of the combustible matter is consumed. The residue is subsequently weighed, and from the amount left the percentage of incombustible matter present is estimated. In making this experiment, much time will be saved by placing the crucible a little on one side, and partially covering its mouth with the lid, for the purpose of directing a current of air on the burning body. When the substance to be examined is a caking coal, it is found advantageous not to break the crust of coke which is first formed, but to allow the mass gradually to consume from the exterior. If this be not attended to, and it should contain much ash, small portions of the fuel are frequently protected by a coating of earthy matter, and escape complete combustion. In the case of coke it is sometimes extremely difficult to consume the last portions of carbon, but this may always be effected either by exposing the crucible and its contents in an assay muffle, or by subjecting the substance, at a red heat, to the action of oxygen gas.

The latter process is accomplished by placing a known weight of pulverized coke in a porcelain crucible over a lamp, and when, from the accumulation of ash, the combustion becomes sluggish, the vessel is closed by a cover having a hole through its centre, and through this a current of oxygen is conducted by a suitable tube, from a gas-holder in which it is contained. The amount employed is regulated by a stop-cock, and too rapid action is especially to be avoided.

HYGROMETRIC WATER.—The amount of water present is estimated by drying a known quantity of the substance in a water or air bath, heated to 100° C., until it ceases to lose weight. In accurate determinations, all experiments should be repeated at least twice, as perfect reliance can never be placed in results when this precaution has not been attended to.

SULPHUR.—The sulphur contained in a fuel is determined by the following process. The substance to be examined is intimately mixed with twice its weight of pure magnesium carbonate, and placed in a bulb blown in the middle of a tube of hard glass. This is heated either by a spirit-lamp or gas-flame, at the same time that a continuous current of oxygen gas is passed through it from an apparatus attached for that purpose. When the whole of the carbonaceous matter has been completely consumed, which is easily perceived by the whiteness of the mixture, the powder is thrown on a filter, and the soluble magnesium sulphate washed through. The sulphuric acid in the filtrate, rendered acid by HCl, is then thrown down by barium chloride, and from the weight of sulphate of barium obtained, the percentage amount of sulphur present is deduced.

Calcium carbonate may be employed for this purpose instead of the corresponding salt of magnesium, but as calcium sulphate is less soluble than sulphate of magnesium, the washing on the filter requires to be prolonged, and a longer time is necessary for the completion of this operation than for that above described.

The sulphur may be more rapidly, but perhaps less correctly, estimated by igniting in a platinum crucible a mixture of the substance to be examined with three times its weight of nitre and four times that quantity of pure sodium carbonate. When this method is employed, the fused mass which remains in the crucible is first dissolved in water, and, after being filtered, is rendered acid by the addition of either nitric or hydrochloric acid. The filtrate is then treated with a sufficient amount of barium chloride as above described.

A paper was read by F. Crace-Calvert at the Edinburgh meeting (1871) of the British Association for the Advancement of Science, on the estimation of sulphur in coal and coke. The sulphur found in coal or coke often exists in two states, partly as sulphuric acid in sulphate of calcium, and partly as sulphur combined with iron.[1] The portion existing as sulphate of calcium does not injure the quality of iron when used in the production of that metal, as it remains associated with calcium; whilst the portion existing as sulphide of iron greatly deteriorates the commercial value of the fuel. To determine the amount of sulphur in the former state, the author proposes to boil the pulverized coke or coal in a solution of sodium carbonate, which decomposes the calcium sulphate or calcium sulphide, and sulphur is estimated in the solution. In the residue from the above operation is found the sulphur combined with iron. After attacking with aqua regia, the author treats with carbonate of sodium and heats to near the fusing point. By this means there can be no formation of insoluble basic sulphate of iron, and the prevention of precipitation by a salt of barium, stated to occur in liquids containing free nitric acid, is avoided.

Carbon and Hydrogen.—These constituents are estimated according to methods employed for the analysis of organic substances; but the best results are obtained when a quantity of matter not exceeding half a gramme is operated on. Experience also shows that the combustion of fuels is more completely effected by cupric oxide than when lead chromate is employed, and that whenever the substance burns with difficulty, as in the case of coal, and more particularly of anthracite, it is necessary, not only to use a long combustion-tube, but also a certain portion of dry chlorate of potassium, which, after being mixed with cupric oxide, is placed at the sealed end of the tube, so as to give off oxygen towards the close of the operation. Instead of using chlorate of potassium, oxygen from a gas-holder may be passed through the tube.

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[1] Sulphur also, doubtless, occurs in combination with the organic elements of coal.

NITROGEN.—The amount of nitrogen contained in a fuel cannot be considered of much practical importance, and it may therefore, in most instances, be included with the oxygen: if necessary, however, it can be estimated as ammonia by the method of Will and Varrentrapp.

OXYGEN.—This element is invariably estimated by the loss on analysis.

Dr. Percy calls attention, as follows, to certain sources of error in analyses of coal:[1] "When coal contains much inorganic matter, especially iron pyrites, the usual method of calculating its composition, from the data obtained in the process of organic analysis, may be erroneous in a sensible degree. The ashes left by incineration are estimated as inorganic matter, and the proportion of oxygen is found by subtracting the sum of the carbon, hydrogen, nitrogen, and ashes, from the amount of dry coal subjected to analysis. By incineration the iron of the pyrites is converted into sesquioxide, and the sulphur, in a greater or less degree, into sulphuric acid, which may remain in combination with any base in the ashes, such as lime, capable of forming a sulphate not decomposable at a red heat. Supposing the whole of the sulphuric acid to be thus retained in the ashes, for 1 part by weight of iron pyrites there would be an increase of 1 part by weight, due to oxygen derived from the air during incineration. The whole amount of this error, provided no correction be made, would fall upon the oxygen. It is not asserted that the whole of the sulphur is actually converted into sulphuric acid and retained in the ashes; but that a considerable portion of a stable sulphate may be produced during incineration will appear from analyses of coal in the sequel. It is certain that the alumina in the ashes must, either wholly or in great measure, exist in combination with silica as clay, but clay holds water in combination which cannot be expelled except at a temperature far more than sufficient to decompose coal. Hence, during the process of organic analysis, water may be evolved from the clay present in coal and coal ashes, and so occasion an error of excess in the determination of the hydrogen. This source of error has been pointed out by Regnault. Carbonate of lime is sometimes present in coal in very appreciable quantity, in which case carbonic acid would be evolved during the analysis, and so an error of excess would be caused in the determination of the carbon. M. de Marsilly has observed, that however pure a piece of coal may be, and however homogeneous it may appear to the eye, its different parts do not yield the same proportion of fixed residue by incineration; and the same is true in respect to the proportion of coke obtained by the calcination of different fragments of the same lump of coal. Hence, in every case, the proportion of ash and coke should be determined by operating upon an average sample taken from the powder of a considerable quantity of the coal."

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[1] Percy's 'Metallurgy;' Fuel, p. 301.

**LITHARGE EXPERIMENTS.**—In some cases, the calorific value of fuels may be ascertained with sufficient accuracy without having recourse to an elaborate analytical examination, and for this purpose the process employed by Berthier is the most simple.

The weight of substance operated on may be about half a gramme, and great care should be taken to obtain it in the finest possible state of division. If the substance be brittle, such as coal, coke, or charcoal, it is easily pounded in a mortar, and afterwards sifted; but if it be a variety of wood which is to be experimented on, the dust obtained by cutting it with a fine file should be employed. This should, according to its supposed richness, be intimately mixed with about 25 grammes of litharge, and placed in an earthen assay crucible. On this is placed from 14 to 15 grammes of pure litharge, and after the whole has been shaken down, the crucible ought not to be more than half full, in order to allow sufficient space for the intumescence of the mixture when in a semi-fluid state. The crucible is now stopped by a cover, which is luted with fire-clay for the purpose of preventing reducing gases from the fire, or fragments of coke, from vitiating the result; and the whole is placed in an assay furnace already lighted, in which there is a supply of hot coke. Here it is gently heated during about fifteen minutes, at the expiration of which time the contents will be in a state of tranquil fusion. The crucible is now to be covered with coke, and the draught increased by means of the damper, in order to cause the whole of the reduced lead to collect in a button at the bottom. Care should likewise be taken to prevent any loss of metal by volatilization, and a moderate temperature only should consequently be employed. This collection of the metal will usually be effected in about ten minutes, and the crucible is then withdrawn from the fire and slightly tapped against some hard body, to throw down any globules which may still remain suspended in the fused litharge. After being allowed to get cold, the crucible is broken and the button of lead extracted, and from its weight is estimated the calorific value of the fuel. If the operation has been properly conducted, the lead separates easily both from the crucible and from the melted litharge; but in case of anything adhering to it, its removal is readily effected by first hammering the button on an anvil, and afterwards brushing off the small particles sticking to it with a hard brush. The results thus obtained from different experiments on the same substance will be found to agree closely with one another. On comparing, however, the calorific value of a fuel, as obtained by the litharge process, with that calculated from its ultimate analysis, the former is sometimes found to be as much as one-ninth less than that obtained by the latter method. This process, therefore, although not admitting of great accuracy, is sufficiently exact for many practical purposes.

The exactitude of such determinations is sometimes also slightly influenced by the presence of iron pyrites and protosulphide of iron, both of which exercise a reducing influence on litharge similar to that produced by carbon and hydrogen. When heated with this substance, the sulphur escapes in the form of sulphurous anhydride, whilst the iron with which it was combined remains with the litharge in the state of oxide. These reactions determine the reduction of a certain quantity of metallic lead, which interferes with the experiments, and, to a certain degree, vitiates the result.

With few exceptions the operations employed for the extraction of metals from their ores require the aid of elevated temperatures, and consequently it is important that the metallurgist should be acquainted with the properties of the various kinds of fuel, and be enabled to judge under what conditions each may be most economically employed.

## WOOD.

Wood consists essentially of organic tissue composed of carbon, hydrogen, and oxygen, with a minute quantity of nitrogen, and a small proportion of inorganic matter. In its ordinary state it contains large quantities of water, which may be completely expelled at a temperature considerably below that at which decomposition of the organic matter would take place.

The elementary composition of the tissue of all wood is the same, although various organic compounds with which it is associated may be very different in trees of different species, *e.g.*, fir trees contain turpentine and oaks tannin. The value of wood as a fuel is almost entirely due to its vascular tissue, since the associated organic compounds are usually too small in quantity to afford calorific effects of any practical importance.

The proportion of water differs in various kinds of woods, and is also considerably affected by the season of the year at which the different specimens may have been felled. When trees are cut during the winter months, and therefore not in a state of active vegetation, the amount is found to be less than if felled in summer when full of sap; and consequently all woods (unless there be some special reason for not doing so) should be cut during the colder portions of the year.

Some kinds of trees are cultivated, not only for the timber which they yield, but also on account of the tannin contained in their bark; and such species are usually cut during the flow of the sap, as they are at that time more easily barked, and likewise contain a larger proportion of the material for the sake of which their bark is collected.

Small shoots and twigs yield a larger percentage of water than the more solid stem; the difference is also very considerable in woods of different botanical species, as may be observed by inspection of the following numbers given by Schübler and Hartig.

One hundred parts of freshly cut wood from the

| | Water. | | Water. |
|---|---|---|---|
| Hornbeam (*Carpinus betulus*) contain | 18·6 | White Fir (*Pinus abies*) contain | 37·1 |
| Willow (*Salix caprea*) | 26·0 | Pine (*Pinus sylvestris*) | 39·7 |
| Sycamore (*Acer pseudoplatanus*) | 27·0 | Beech (*Fagus sylvatica*) | 39·7 |
| Mountain Ash (*Pyrus aucuparia*) | 28·3 | Alder (*Betula alnus*) | 41·6 |
| Ash (*Fraxinus excelsior*) | 28·7 | Aspen (*Populus tremula*) | 43·7 |
| Birch (*Betula alba*) | 30·8 | Elm (*Ulmus campestris*) | 44·5 |
| Wild Service Tree (*Cratægus torminalis*) | 32·3 | Silver Fir (*Pinus picea*) | 45·2 |
| Oak (*Quercus robur*) | 34·7 | Lime Tree (*Tilia europæa*) | 47·1 |
| Pedicle Oak (*Q. pedunculata*) | 35·4 | Italian Poplar (*Populus dilatata*) | 48·2 |
| Horse-chestnut (*Æsculus hippocastanum*) | 38·2 | Larch (*Pinus larix*) | 48·6 |
| | | White Poplar (*Populus alba*) | 50·6 |
| | | Black Poplar (*Populus nigra*) | 51·8 |

It follows that recently cut wood contains from one-fifth to one-half its weight of water, which not only detracts from its value as a fuel in the same proportion, but, from its escaping in the form of vapour, must, moreover, carry off a portion of the heat developed by the combustion of the other constituents.

By exposure to the air green wood soon loses part of its water, but after a time ceases to diminish in weight, a sort of equilibrium being established between the hygroscopic power of the air and that of the wood. When this occurs, no further drying is effected by continued exposure, and its percentage of water will vary only within very narrow limits, dependent on the dryness or humidity of the situation in which it may be placed.

In this state wood is said to be *air-dried*, and the remaining portions of moisture can only be expelled by the aid of heat, the last traces being eliminated with extreme difficulty.

Rumford, who heated specimens of various kinds of air-dried woods at a temperature of 136° C. until they ceased to lose weight, obtained the following results:—

One hundred parts of

| | | | |
|---|---|---|---|
| Oak wood lost | 16·64 | Fir wood lost | 17·53 |
| Elm | 18·20 | Birch | 19·38 |
| Beech | 18·56 | Lime | 18·79 |
| Maple | 18·63 | Poplar | 19·55 |

Generally speaking, the wood employed for fuel is never thoroughly dried, but retains from 15 to 25 per cent. of water, so that the driest specimens seldom contain more than 85 per cent. of combustible matter. Wood kept in a warm room for six months still retains, according to Winkler, about 17 per cent. of water. Woods are usually divided into two classes—*hard* and *soft*. This distinction is founded on their calorific properties and on the facility with which they can be worked by edge-tools.

The former, among which are numbered oak, beech, white and red birch, and elm, contain in the same bulk a larger proportion of fibre, and have their vessels more closely packed, than those of the softer varieties, such as pine, fir, larch, lime, willow, and the various kinds of poplar.

Trees which have grown in poor land and in exposed situations, are supposed to produce harder and denser wood than individuals of the same kind which have been planted in more sheltered localities and in richer soils. The specific gravity of wood depends in a great measure on its structure; but as two specimens of the same tree are never found perfectly homogeneous, the results obtained by experiment should rather be considered as approximate than as representing the true density of the wood examined. These variations of specific gravity will also be influenced, to a certain extent, by the nature of the soil in which the tree has grown, as on this will depend, in a great measure, the quantity and character of the salts it contains.

From the air enclosed in their cavities, woods are in their ordinary state generally lighter than the same bulk of water; but when reduced to the state of fine powder, even the softest varieties are found to possess a greater density than that liquid. By thus destroying the pores and liberating the enclosed air, the specific gravity of the following woods has been found to be—

| | | | |
|---|---|---|---|
| Oak | 1·27 | Fir | 1·16 |
| Lime | 1·13 | Beech | 1·29 |

In the following table are given the respective densities of different kinds of wood in various states of dryness:—

SPECIFIC GRAVITY OF DIFFERENT KINDS OF WOOD.

| Variety of Wood. | 1. Recently Felled. | 2. Dried in Air. | 3. Strongly Dried. |
|---|---|---|---|
| Common Oak (*Quercus robur*) | 1·0754 | 0·7075 | 0·663 |
| Pedicle Oak (*Q. pedunculata*) | 1·0494 | 0·6777 | 0·663 |
| White Willow (*Salix alba*) | 0·9859 | 0·4873 | 0·457 |
| Beech (*Fagus sylvatica*) | 0·9822 | 0·5907 | 0·560 |
| Elm (*Ulmus campestris*) | 0·9476 | 0·5474 | 0·518 |
| Hornbeam (*Carpinus betulus*) | 0·9452 | 0·7695 | 0·691 |
| Larch (*Pinus larix*) | 0·9205 | 0·4735 | 0·441 |
| Pine (*Pinus sylvestris*) | 0·9121 | 0·5502 | 0·485 |
| Sycamore (*Acer pseudoplatanus*) | 0·9036 | 0·6592 | 0·618 |
| Ash (*Fraxinus excelsior*) | 0·9036 | 0·6440 | 0·619 |
| Birch (*Betula alba*) | 0·9012 | 0·6274 | 0·598 |
| Mountain Ash (*Pyrus aucuparia*) | 0·8993 | 0·6440 | 0·552 |
| Fir (*Pinus abies*) | 0·8941 | 0·5550 | 0·493 |
| Silver Fir (*Pinus picea*) | 0·8699 | 0·4716 | 0·434 |
| Wild Service (*Cratægus torminalis*) | 0·8633 | 0·5910 | 0·549 |
| Horse-chestnut (*Æsculus hippocastanum*) | 0·8614 | 0·5749 | ... |
| Alder (*Betula alnus*) | 0·8571 | 0·5001 | 0·443 |
| Lime (*Tilia europæa*) | 0·8170 | 0·4390 | 0·431 |
| Black Poplar (*Populus nigra*) | 0·7795 | 0·3656 | 0·346 |
| Aspen (*Populus tremula*) | 0·7654 | 0·4302 | 0·418 |
| Italian Poplar (*Populus dilatata*) | 0·7634 | 0·3931 | ... |
| Ebony (*Diospyrus ebenum*) | ... | 1·2260 | ... |

Columns 1 and 2 give the densities determined by Hartig, and column 3 the results obtained by Winkler, who weighed a cubic inch of each kind of wood.

Wood not only loses weight by exposure to air, but at the same time decreases in bulk; and in some varieties this takes place to the extent of one-tenth of its original volume.

By long immersion in water, the soluble and extractive matters contained in woods are dissolved, and therefore the method of transporting it by rafts, as practised in some countries, is not only found to lessen its weight, but also to reduce its calorific power; consequently the advantages of cheap transport are to some extent counterbalanced by the inferiority of the wood so conveyed.

The analysis of different kinds of wood yields results differing little from one another; but in all the varieties yet examined there is a slight excess of hydrogen over oxygen, although in pure woody fibre they are combined in such proportion as, by their union, to form water.

The following table, compiled from the results of Petersen and Schödler, and from those of Heintz, gives the elementary composition of different varieties of dry wood. The specimens analysed were in each case taken from the trunk. Dr. Percy, who gives these and numerous other analyses of wood, calls attention to the fact that a small amount of error must exist in all of them, from the presence of a small quantity of $CO_2$ in the ashes; but remarks that it cannot exceed 0·2 per cent.[1]

ELEMENTARY COMPOSITION OF DRY WOOD.

| Name of Tree. | Exclusive of Ash. | | | |
|---|---|---|---|---|
| | C. | H. | O. | N. |
| Oak | 48·94 | 5·94 | 43·09 | 2·03 |
| Beech | 48·29 | 6·00 | 45·14 | 0·57 |
| Birch | 48·89 | 6·19 | 43·93 | 0·99 |
| Hornbeam | 48·08 | 6·12 | 44·93 | 0·87 |
| Alder | 48·63 | 5·94 | 44·75 | 0·68 |
| Ash | 49·36 | 6·08 | 44·56 | ... |
| Horse-chestnut | 49·08 | 6·71 | 44·21 | ... |
| Black Poplar | 49·70 | 6·31 | 43·99 | ... |
| Lime | 49·41 | 6·86 | 43·73 | ... |
| Scotch Fir, Old | 49·87 | 6·09 | 43·41 | 0·63 |
| " Young | 50·62 | 6·27 | 42·58 | 0·53 |
| Spruce Fir | 49·95 | 6·40 | 43.65 | ... |
| Walnut | 49·12 | 6·44 | 44·44 | ... |
| Mean | 49·22 | 6·25 | 44·02 | 0·45 |

The nature and amount of ash left by the combustion of the various kinds of wood, depend, not only on the species of tree examined, but are also, to a certain degree, influenced by the nature of the soil on which it has been produced, as the different inorganic substances which enter into the composition of plants seem to have, to some extent, the power of replacing one another. Generally speaking, the ash of wood contains potassium, sodium, magnesium, and iron as carbonates, silicates, sulphates, phosphates, and chlorides.

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[1] Percy's 'Metallurgy;' Fuel, p. 185.

The following table gives the amount of ash remaining after the combustion of different varieties of wood:—

| | Percentage of Ash. | | Percentage of Ash. |
|---|---|---|---|
| Fir | 0·83 | Elder | 1·64 |
| Birch | 1·00 | Lime | 1·45 |
| Hazel | 1·57 | Oak, branches | 2·50 |
| White Mulberry | 1·60 | „ bark | 6·00 |
| Saint Lucia | 1·60 | | |

The different parts of the same tree do not yield equal proportions of incombustible matter; the bark and leaves always produce a larger amount than the branches, whilst the branches leave more than the trunk. Woody plants generally yield less ash than herbaceous ones, which are also remarkable for containing a larger proportion of silica than is usually met with in wood.

Many of the published analyses of the ashes of wood are either imperfect or otherwise unsatisfactory. The following, made by Böttinger under the direction of Will, appear to be among the most complete and reliable. In these analyses the calculations have been made after deduction of $CO_2$, and the charcoal resulting from imperfect incineration.[1]

Composition of the Ashes of Wood.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| $K_2O$ | 15·80 | 2·79 | 0·93 | 15·24 |
| $Na_2O$ | 2·76 | 15·99 | 14·59 | 7·27 |
| $CaO$ | 60·35 | 30·36 | 33·99 | 25·85 |
| $MgO$ | 11·28 | 19·76 | 20·00 | 24·50 |
| $Mn_3O_4$ | ... | 18·17 | 7·61 | 13·51 |
| $Fe_2O_3$, $P_2O_5$, or $(Fe''' PO_4)$ | 1·84 | 5·10 | 2·28 | 6·18 |
| $Fe_3O_4$ | ... | ... | 7·73 | ... |
| $3\ CaO$, $P_2O_5$, or $Ca''_3\ (PO_4)_2$ | 3·99 | ... | ... | ... |
| $CaSO_4$ | 2·30 | 3·31 | 5·05 | 2·91 |
| $NaCl$ | 0·21 | 1·48 | 2·52 | 0·92 |
| $SiO_2$ | 1·46 | 3·04 | 5·27 | 3·60 |
| | 99·99 | 100·00 | 99·97 | 99·98 |

No. 1, beech from Switzerland; Nos. 2 and 3, Scotch fir, from the neighbourhood of Giessen, near which there are mines of manganese; No. 2 from a diseased tree, and No. 3 from one that had died; No. 4, larch from the same locality as Nos. 2 and 3.

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[1] 'Annalen der Chemie und Pharmacie,' vol. 50 p. 406, 1844.

## PEAT OR TURF.

In low and moist situations, where water collects and cannot readily flow off, and in which the loss by evaporation is inconsiderable, large swamps or bogs are formed, and in these marsh-plants of all kinds, such as sedges, rushes, reeds, mosses, confervæ, and even small shrubs, grow with great rapidity, and quickly cover the surface with a thick layer of vegetation. In winter these die, and are, on the return of spring, themselves covered by another crop of similar plants. These changes go on from year to year, and finally the ground becomes covered by a thick layer of vegetable matter in a loose state of aggregation. After a time, decomposition takes place in the mass, carbonic anhydride and marsh-gas, together with small quantities of sulphuretted hydrogen (produced by the reduction of sulphates), are evolved, and finally the whole attains a considerable density and becomes of a dark earthy colour.

This substance, which is called *peat*, is in many places extensively employed as fuel. There are but few countries in which small areas of this material are not found; but in some, such as in Holland and in North Germany, such formations extend over districts of immense area, and annually furnish large quantities of fuel.

Sometimes these deposits appear to have taken place at successive periods, and in such cases they are generally divided into parallel horizontal strata by layers of earth or sand of varying thickness. The beds nearest the surface are for the most part less compact, and of a lighter colour, than those lying deeper in the series, and are made up of the roots and stems of plants, which, although to some extent decomposed, still retain their original forms.

This porous spongy substance is called *turf*, and usually becomes of a darker colour and greater density as its depth increases; finally, it loses all outward traces of its vegetable origin, and is transformed into the dark substance called peat.

Peat is turf so far decomposed that traces only of its original organic structure remain, and of which the fracture has become compact, and in some instances even resinous. Its density is also greater than that of the more recent variety, of which a cubic foot only weighs from four to six pounds, while the weight of the same bulk of ordinary peat varies from twelve to twenty pounds.

The cutting of peat is a simple operation. After having laid bare the surface, the peat is cut by square-pointed shovels into the shape of rectangular blocks, which are usually dried in the sun, and subsequently stacked as fuel. In some instances the surface of the ground is covered with water, which cannot be drawn off. In such cases the peat is collected by means of an instrument resembling a square-pointed shovel provided with an edge turned up at right angles for the purpose of affording a hold for the block after its separation from the mass. To use this tool, a man stands on a stage slightly above the surface of the water, and having thrust the instrument into the peat, withdraws it, together with a rectangular prism of the combustible, attached by adhesion to its two sides. When the depth of the water is more considerable, a larger instrument is employed, which is worked by two men, and provided with a spring for holding the detached prism with sufficient firmness to allow of its being drawn to the surface, where the spring is released and the charge withdrawn.

In Holland and elsewhere, when peat is too spongy to be extracted by the method above described, and has become reduced to the state of black mud, it is obtained by the use of a dredge, made of a sharp steel hoop, to which is attached a bag of close network, which allows the water to flow through, but retains the particles of peaty matter scraped from beneath the surface of the water. These are allowed to drain in wooden troughs, the bottoms of which are covered with straw, and in which numerous holes are bored for the purpose of allowing the escape of water. When the mass has thus attained a certain consistence, it is trodden down by persons wearing large pieces of wood on their feet like snow-shoes, to prevent their sinking into it; and, when sufficiently firm to resist the pressure of the foot, it is beaten with a beater of peculiar construction until a large portion of the water has been expelled. The mass is now cut into blocks not unlike bricks, which are stacked under proper sheds, so as to allow currents of air to pass between the different layers, thereby facilitating the drying of the blocks.

The amount of water contained in air-dried peat varies considerably, but is usually from 15 to 20 per cent. At 120° C. peat begins to suffer decomposition, and when heated to 250° C., not unfrequently ignites. The specific gravity of uncompressed peat varies from 0·25 to 1·058.

### Elementary Composition of Dry Peat.

| Locality. | Exclusive of Ash. | | | Analysts. |
|---|---|---|---|---|
| | C. | H. | O. & N. | |
| Cappoge, Ireland . | 52·38 | 7·03 | 40·59 | Kane. |
| Kilbeggan " . | 62·18 | 6·79 | 31·03 | |
| Kilbaha " . | 55·62 | 6·88 | 37·50 | |
| Vulcaire, France . | 60·40 | 5·95 | 33·65 | Regnault. |
| Long, " . | 60·90 | 6·22 | 32·88 | |
| Champ-du-Feu " . | 61·05 | 6·45 | 32·50 | |
| Mean . . | 58·75 | 6·56 | 34·69 | |

The ashes which remain after burning peat are partly due to the salts contained in the plants from which it derived its origin; but the larger proportion arises from earthy matters subsequently deposited from the waters which so frequently cover the surface of the bogs on which it is produced.

The composition of the ashes of peat will necessarily, to a great extent, be influenced by the nature of the soil of the district in which it was formed, as the water descending from higher grounds during heavy rains will always carry with it, in suspension, small particles, which, on reaching the lower lands, are deposited in the form of sand or mud. In fact, it is constantly observed that the ashes of peat from a calcareous region contain large quantities of lime, while specimens which have been formed amongst quartzose rocks yield an ash in which siliceous materials predominate.

In 100 parts of peat the following quantities of ash have been observed:—

| Variety of Peat. | Ash. | Observers. |
|---|---|---|
| Grass peat, brownish yellow . . . . . . | 17·30 | Berthier. |
| Pitch peat, from Clermont . . . . . . | 26·00 | |
| Herbaceous, from Burgundy . . . . . | 7·10 | |
| Brown and herbaceous, from Troyes . . . | 16·00 | |
| Very old peat, from Vulcaire, near Abbeville . | 5·88 | Regnault. |
| „ „ Long . . . . . | 4·61 | |
| Not so old, from Champ-du-Feu, Vosges . . | 5·35 | |
| Near Berlin, 1st stage . . . . . . . | 9·30 | Achard. |
| „ „ 2nd „ . . . . . . . . | 10·20 | |
| „ „ 3rd „ . . . . . . . . | 11·20 | |
| Moor in Eichsfeld, 1st sort . . . . . | 21·50 | Buchholz. |
| „ „ 2nd „ . . . . . . . | 23·0 | |
| „ „ 3rd „ . . . . . . . | 30·5 | |
| „ „ 4th „ . . . . . . . | 33·0 | |
| Yellowish-brown, from Dartmoor . . . . | 13·43 | J. A. Phillips. |

Peat can only be advantageously employed by the metallurgist in localities where other fuels are scarce and expensive, as the great space which it occupies, the large percentage of moisture retained by it, the difficulty of drying the requisite quantities, and the amount of ash resulting from its combustion, present obstacles to its general use which are not readily overcome. Various processes have, at different periods, been employed to improve the quality of peat, but, up to the present time, none of them appear to have afforded results that can be regarded as entirely satisfactory. In some processes the blocks, after being partially dried by exposure to sun and air on the surface of the ground, are stacked in hollow piles, and finally placed in kilns, through which currents of heated air are conducted. In others the peat is first worked up into a pulpy mass in pug-mills, and then moulded by machinery into blocks, which are afterwards kiln-dried. By these means the quality of the fuel can be materially improved, but the cost of labour and machinery is so considerable, that the product cannot, under ordinary circumstances, compete with other varieties of fuel. It is, however, probable that an increase in the price of coal may at some future time lead to the discovery of means by which the large amount of combustible matter locked up in the peat-beds of this and other countries, may be rendered extensively available.

The following analyses of the ashes of peat have been selected from a series of twenty-seven made, under the direction of Sir Robert Kane, in the laboratory of the Museum of Irish Industry :—

COMPOSITION OF THE ASHES OF PEAT.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| $K_2O$ . . . . . . . | 0·362 | 0·641 | 0·744 | 1·667 |
| $Na_2O$ . . . . . . | 1·427 | 1·875 | 0·704 | 2·823 |
| $CaO$ . . . . . . . | 26·113 | 22·702 | 40·623 | 20·907 |
| $MgO$ . . . . . . | 3·392 | 6·809 | 4·352 | 5·252 |
| $Al_2O_3$ . . . . . . | 4·180 | 1·109 | 1·671 | 2·034 |
| $Fe_2O_3$ . . . . . . | 11·591 | 29·854 | 10·368 | 17·040 |
| $P_2O_5$ . . . . . . | 1·461 | 2·019 | 1·114 | 1·447 |
| $SO_3$ . . . . . . . | 12·403 | 16·381 | 24·208 | 23·375 |
| $HCl$ . . . . . . . | 1·568 | 1·591 | 1·052 | 1·424 |
| $SiO_2$, in compounds decomposable by acids . . . . | 0·980 | 0·737 | 6·317 | 6·634 |
| Sand and silicates undecomposable by acids . . . | 22·519 | 14·505 | 3·710 | 10·682 |
| $CO_2$ . . . . . . . | 13 695 | 1·470 | 4·981 | 6·721 |
| | 99·691 | 99·693 | 99·844 | 100·006 |

1. Light spongy surface peat, of a reddish-brown colour, and composed almost entirely of *Sphagnum*, pieces of which are still distinguishable; from near Monastereven.

2. Good compact peat, of a blackish-brown colour, consisting principally of moss, with a number of *Erica* and grass roots, with *Carex*. This peat, which is used as fuel in Dublin, is from Riversdale Bog, near Kinnegad.

3. An exceedingly dense peat, with a conchoidal earthy fracture; from Athlone bog. Vegetable structure almost obliterated, but, when apparent, indicates remains of *Carex*, grasses, and *Erica* in abundance.

4. A rather dense peat, of a blackish-brown colour, in which the structure of moss is no longer visible, but abounding in remains of *Carex*, grasses, and roots and stems of *Erica;* from the Curragh or Clonburne bogs, near Shannon Bridge.

In almost every case the amount of $CO_2$ found was considerably less than that required for the formation of calcium carbonate, after admitting the whole of the sulphuric acid to exist as calcium sulphate. This was supposed to have been caused by the expulsion of $CO_2$ during the process of incineration. Regnault, however, satisfied himself that the whole of the calcium present in peat does not exist as carbonate, but to some extent as forming part of various organic compounds.

The following analyses of the ashes of American peat are given by Professor Johnson:—[1]

Composition of the Ashes of Peat—American.

| | 1. | 2. | 3. |
|---|---|---|---|
| $K_2O$ | 0·69 | 0·80 | 3·46 |
| $Na_2O$ | 0·58 | ... | trace |
| $CaO$ | 40·52 | 35·59 | 6·60 |
| $MgO$ | 6·06 | 4·92 | 1·05 |
| $Fe_2O_3$ and $Al_2O_3$ | 5·17 | 9·08 | 15·59 |
| $P_2O_5$ | 0·50 | 0·77 | 1·55 |
| $SO_3$ | 5·52 | 10·41 | 4·04 |
| Cl | 0·15 | 0·43 | 0·70 |
| $SiO_2$, soluble | 8·23 | 1·40 | 67·01 |
| $CO_2$ | 19·60 | 22·28 | |
| Sand | 12·11 | 15·04 | |
| | 99·13 | 100·72 | 100·00 |

The specimens of peat affording the ashes analysed were obtained from three different localities in the State of Connecticut. 1. From Poquonnock; analysed by Professor G. F. Barker. 2. From Colebrook; by Mr. O. C. Sparrow. 3. From Guildford; analysed by Mr. Peter Collier.

Karsten states that peat sometimes contains iron pyrites in sufficient quantity to admit of its being employed for the manufacture of green vitriol. In a deposit of peat occurring near Moel-Hafod-Owen, North Wales, copper was some years since found in sufficient quantities to repay the expenses of working it for that metal. For this purpose the peat was burnt in kilns, and the ashes subsequently collected for smelting. At the present time the whole of this cupriferous peat has been removed, but the water, which sometimes collects in pools on the surface, is still often tinged with green from the presence of copper salts. These are probably derived from the drainage of a large vein, largely composed of arsenical pyrites, which traverses the hill at the bottom of which the peat was found.

## COAL.

Coals constitute an important family, embracing *lignite* or *brown coal*, *common* or *bituminous coal*, and *anthracite*. The chief constituent of coal is carbon, in chemical combination with varying proportions of hydrogen, oxygen, and nitrogen. All coals contain a greater or less amount of earthy impurity, which being incombustible, remains, after burning, in the form of ash. From their composition, structure, and other characteristics, there can be no doubt of the vegetable origin of coals; whether occurring as lignite, in which the woody structure is still apparent, or as common coal or anthracite, in which mineralization has, for the most part, so far advanced that all external traces of their organic derivation have been obliterated. We have conclusive evidence that coal of every species is merely mineralized vegetation, which, in part, grew *in situ* as peat-mosses, cypress-swamps, forest-growths, &c., and was, in part, *drifted* by the action of water to basins of deposit. The operations of nature being incessant, we have coals belonging to all periods,—peats of the current epoch, lignites of the Tertiary, bituminous coals of the Carboniferous, anthracites and graphites of the Devonian and Silurian, &c. These products differ from one another according to the amount of metamorphism to which they may have been severally subjected. The coal of Great Britain is of Carboniferous age, but many productive coal-fields in India, America, and elsewhere belong to the Jurassic and Cretaceous periods. Coals are sometimes so free from earthy matter as to leave less than one per cent. of ash; whilst others, on the contrary, are so impure as to be unfit for fuel, and thus pass into bituminous shales.

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[1] 'Peat and its Uses as Fertilizer and Fuel,' by S. W. Johnson, M.A., p. 47. New York.

The formation of the different varieties of coal by the decomposition of woody tissue may be explained by the gradual elimination of hydrogen and carbon as marsh-gas, of oxygen and carbon as carbonic anhydride, and of oxygen and hydrogen in the form of water. That the transformation of woody tissue into coal has been accompanied by the production of marsh-gas may be inferred from the composition of the fire-damp of coal mines, of which this gas is the chief constituent. It is not difficult to select from analyses of fuels a series illustrating the gradual passage of woody tissue into anthracite; a coal consisting almost exclusively of carbon. The following table, arranged by Dr. Percy, gives the variable amounts of hydrogen and oxygen contained in different kinds of fuel; the amount of carbon in each case being represented by a constant quantity.[1]

| | C. | H. | O. |
|---|---|---|---|
| Wood (mean of twenty-six analyses) | 100 | 12·18 | 83·07 |
| Peat | 100 | 9·85 | 55·67 |
| Lignite (average of fifteen varieties) | 100 | 8·37 | 42·42 |
| Ten-Yard Coal, South Staffordshire | 100 | 6·12 | 21·23 |
| Steam Coal, from the Tyne | 100 | 5·91 | 18·32 |
| Pentrefelin Coal, South Wales | 100 | 4·75 | 5·28 |
| Anthracite, Pennsylvania, U.S. | 100 | 2·84 | 1·74 |

It will be observed that throughout the various stages of conversion the relative proportions of hydrogen and oxygen gradually decrease.

Nitrogen is present in coal in small proportions (up to about 2 per cent.), and although it does not appear to be an essential constituent of woody fibre, yet all woods contain albumen, and other nitrogenous compounds, which readily account for the presence of this element in the different varieties of fossil fuel.

Sulphur is always present in coal, in which it exists chiefly in the form of iron pyrites; it may also occur as calcium sulphate, and also combined with the organic elements of coal. Coal always contains a certain proportion of water, which may be expelled at a temperature slightly above 100° C. Whether the whole of this exists in the state of hygroscopic water, or whether, in some cases, a portion of it may not be present in a state of combination, has not been determined. A coal may appear to be perfectly dry, and yet lose a large percentage of water by desiccation.

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[1] Percy's 'Metallurgy;' Fuel, p. 268.

A sensible amount of inorganic matter is contained in all coal; its constituents are chiefly silica, alumina, lime, and oxide of iron, which are, in part, derived from the plants from which the coal was produced, and partly from the percolation of waters holding these substances either in solution or suspension. These substances constitute the ashes which are left by coal when burnt, and its value as a fuel is considerably influenced, not only by the amount of its ashes, but also by their composition. Iron pyrites in coal is represented in the ash by oxide of iron, which, when present in large proportion, has the effect of rendering it fusible and causing it to form clinkers which adhere firmly to the furnace-bars. The distinction between *red* and *white ash* coals is mainly dependent on the amount of pyrites they respectively contain. Daubrée detected arsenic in the Tertiary lignite of Lobsann, Lower Rhine; and galena, copper pyrites, and micaceous iron ore are sometimes met with in this country in coals of Carboniferous age.

LIGNITE OR BROWN COAL.—Lignites consist of fossil plants more or less completely mineralized and converted into coal; they have usually a dull dark brown colour, are compact or laminated, and generally reveal the texture of wood. When burnt, they afford a dull flame and evolve much smoke; they are poorer in carbon than ordinary coal; their heating power is less, and, in the majority of cases, they contain a large amount of ash. Beds of lignite sometimes occur in the New Red Sandstone, Oolite, and Cretaceous formations, but chiefly in the Tertiary. Lignites present a great variety of aspects; some being almost as hard as true coal, while others are distinctly woody. Some, again, consisting of thin layers like compressed leaves, are called "paper coal;" whilst soft earthy varieties have received the name of "peat coal."

Lignite thus passes through every gradation of texture, from that of the more compact peats of the present day to that of the bituminous coals of the older formations. The well-known lignites or *brown coals* of the Continent of Europe are chiefly of Tertiary age, and, from the leaves, fruit, and stems of palms, &c., which they contain, give evidence of the prevalence of a genial climate in these latitudes during that period.

According to Frémy, lignites may be distinguished from mere wood and peat, on the one hand, by their solubility in nitric acid and hypochlorites, and from true coal, on the other, which is insoluble in hypochlorites, and only slowly attacked by nitric acid.

The table on the opposite page shows the percentage composition of different varieties of lignite or brown coal.

COMPOSITION OF LIGNITES, DRIED AT SLIGHTLY ABOVE 100° C.

| No. | Locality. | Specific Gravity. | C. | H. | O. | N.[1] | S. | Ash. | Water, per Cent. | Coke, per Cent. | Exclusive of Sulphur, Ash, and Water.[2] | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | | | C. | H. | O. inclusive of N. |
| 1. | Bovey, Devonshire | 1·13 | 66·31 | 5·63 | 22·86 | 0·57 | 2·36 | 2·27 | 34·66 | 30·79 | 69·53 | 5·91 | 24·56 |
| 2. | Thallern, Austria | 1·41 | 49·58 | 3·84 | 22·68 | ... | 4·56 | 19·84 | 22·53 | 63·70 | 65·15 | 5·05 | 29·80 |
| 3. | Gloggnitz „ | 1·36 | 57·71 | 4·49 | 22·14 | ... | 3·12 | 12·54 | 25·15 | 54·40 | 68·42 | 5·33 | 26·25 |
| 4. | Riestedt, Prussia | 1·21 | 61·13 | 5·09 | 31·95 | ... | ... | 1·83 | 31·66 | ... | 62·27 | 5·18 | 32·55 |
| 5. | Löderburg „ | 1·22 | 55·30 | 4·90 | 31·95 | ... | ... | 7·85 | 49·50 | ... | 60·01 | 5·31 | 34·68 |
| 6. | Wittenberg, on the Oder | ... | 64·07 | 5·03 | 27·55 | ... | ... | 3·35 | 17·26 | ... | 66·29 | 5·20 | 28·51 |
| 7. | Hessen-Cassel | ... | 62·60 | 5·02 | 26·52 | ... | ... | 5·86 | ... | ... | 66·49 | 5·33 | 28·18 |
| 8. | Sipplingen, Lake Constance | ... | 61·40 | 3·28 | 29·82 | ... | ... | 5·50 | ... | ... | 64·96 | 3·48 | 31·56 |
| 9. | Island of Sardinia | ... | 59·98 | 4·75 | 29·42 | ... | ... | 5·85 | ... | 50·00 | 63·71 | 5·05 | 31·24 |
| 10. | British North America, | ... | 65·64 | 4·24 | 21·97 | 0·93 | 0·70 | 6·52 | 13·92 | ... | 71·46 | 4·62 | 23·92 |
| 11. | Prairies east of Rocky Mountains | ... | 59·18 | 3·79 | 16·85 | 1·05 | 0·49 | 18·64 | 14·50 | ... | 74·14 | 4·75 | 21·11 |
| 12. | Island of Trinidad | ... | 75·63 | 5·20 | 13·57 | ... | 2·96 | 2·64 | 20·50 | ... | 80·26 | 5·52 | 14·22 |
| 13. | „ „ | ... | 73·11 | 5·63 | 17·08 | ... | 0·57 | 3·61 | 5·90 | ... | 77·16 | 5·95 | 16·89 |
| 14. | Auckland, New Zealand | ... | 64·70 | 4·81 | 18·25 | 1·34 | 0·42 | 10·48 | 14·12 | ... | 73·72 | 5·48 | 20·80 |
| 15. | Tasmania | ... | 69·14 | 5·40 | 18·48 | 1·26 | 0·35 | 5·87 | 13·43 | ... | 74·33 | 5·80 | 19·87 |

[1] When the proportion of nitrogen is not given, it is included with the oxygen.
[2] Except where the amount of sulphur is not given in the eighth column.

No. 1. Brown; structure fibrous and lamellar; becomes rotten by immersion in water; does not soil the fingers; coke has a semi-metallic lustre; does not swell, and cakes but slightly; ash bulky and red; copper and lead were detected in this lignite; on burning, it evolves an extremely offensive odour; analysed by F. Vaux. 2. Black-brown. This lignite, after being dried, absorbed from the atmosphere 12·7 per cent. of water in twenty-four hours. 3. Black-brown, with woody structure. The dry lignite absorbed from the atmosphere 15·9 per cent. of water in twenty-four hours. 2, 3. Analysed by Schrötter. 4, 5. Brown coal from Prussia; 4, presenting wood-like structure, ash reddish-white; 5, earthy, ash yellowish-brown. The specific gravity and water were determined on material fresh from the workings; analysed by F. Bischof. 6. Brown coal; by Baer. 7. Brown coal; analysed by Liebig. 8. Brown coal; by L. Gmelin. 9. Occurs at a short distance from the sea at Goneza, province of Iglesias, west of Cagliari; probably belongs to the true Coal-measures; analysed at the École des Mines, Paris. 10, 11. Brought by Dr. Hector from La Roche Percée, Saskatchewan Plains. 10. Dark brown; compact, in part wood-like, and in part resembling coal from the Coal-measures; fracture more or less conchoidal. 11. Cracked into small pieces by exposure to the air; in appearance much resembling coal from the Coal-measures. 12, 13. Collected by Mr. G. P. Wall in the Island of Trinidad. 12. Black; fracture dull; powder brown; does not cake when heated in a close vessel; yields 43·15 per cent. of a non-coherent coke. 13. Black; bright, like good bituminous coal; when heated evolves an odour resembling that of petroleum. 14. Black; lustre dull; fracture uneven, more or less conchoidal; cleavage distinct; more or less translucent. A brown resin occurs, diffused through this lignite, in pieces varying from the size of a pea to considerable masses. 15. In physical characters this lignite is similar to that last described, and also contains resin distributed throughout its substance. Accompanying the specimen forwarded was a piece of resin as large as the fist, which was more opaque and less resembling ordinary amber than that contained in the lignite from New Zealand. The analyses 10–15, both included, were made by Mr. C. Tookey, under the direction of Dr. Percy. The mode of rendering the results has, in some cases, been slightly changed, in order to adapt them to the general headings of the table.

The following analyses of the ashes of lignite will be sufficient to give an idea of their general composition:—

| | 1. | 2. |
|---|---|---|
| $SiO_2$ . . . | 3·12 | 19·27[1] |
| $Fe_2O_3$ . . . | 32·78 | 5·78 |
| $Al_2O_3$ . . . | 29·50 | 11·57 |
| CaO . . . | 20·56 | 23·67 |
| MgO . . . | 2·16 | 2·58 |
| $K_2O$ . . . | 0·99 | 1·74 |
| $Na_2O$ . . . | 1·72 | ... |
| $SO_3$ . . . | 9·17 | 33·83 |
| $CO_2$ . . . | ... | ·90 |
| | 100·00 | 99·34 |

1. From Artern, Saxony; analysed by Kremers. 2. From Brunswick; analysed by Varrentrapp. The term "lignite" is sometimes employed as synonymous with coal occurring in deposits of later date than the true Coal-measures.

The coals of Monte Diablo, California, are of Upper Cretaceous age, as are also those of Bellingham Bay, and Naniamo, Vancouver. These coals are extensively used on the Pacific coast, and, like all others of a later geological period than the Carboniferous, contain a considerable proportion of hygroscopic water. Exposed to the action of a dry atmosphere they part with their moisture but slowly, and, in doing so, are liable to become disintegrated. This defect is, however, characteristic of certain American coals of Carboniferous age; the percentage of water in some of the Iowa coals being as great as in those of Monte Diablo. In other respects these Cretaceous coals closely resemble the highly bituminous varieties from the Coal-measures.

The following proximate analyses of various Cretaceous coals, in use on the Pacific coast, are given by Prof. J. D. Whitney, who remarks that those of the Monte Diablo coals were made very shortly after the first opening of the mines, and consequently an improvement in quality might be expected at greater depths.[2]

PROXIMATE COMPOSITION OF CRETACEOUS COALS.

| | Monte Diablo. | | | Bellingham Bay, Washington Territory. | Naniamo, Vancouver. |
|---|---|---|---|---|---|
| | Clark & Co. | Black Diamond. | Cumberland. | | |
| Water . . . | 13·47 | 14·69 | 13·84 | 8·39 | 2·98 |
| Bituminous substances | 40·36 | 33·89 | 40·27 | 33·26 | 32·16 |
| Fixed carbon . . | 40·65 | 46·84 | 44·92 | 45·69 | 46·31 |
| Ash . . . . | 5·52 | 4·58 | 0·97 | 12·66 | 18·55 |

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[1] Residue insoluble in acids.

[2] 'Geological Survey of California,' p. 30.

BITUMINOUS COAL.—This term is usually applied to coals from the Coal-measures; these burn with a more or less smoky flame, like that of bitumen, although the presence of this mineral cannot be detected in ordinary bituminous coal. Coals of this description are brittle and opaque, with a lustre varying from dull to shining; colour, under ordinary circumstances, black or brownish-black; when in fine powder, brown-black or brown; fracture uneven or somewhat conchoidal, the fragments often presenting more or less cubical or rhombic forms; consist of carbon, hydrogen, oxygen, nitrogen, and sulphur, with variable amounts of inorganic matter or ash. When heated in a close vessel they leave a solid carbonaceous residuum, termed coke. Bituminous coals are divided into various classes in accordance with their peculiar chemical and physical properties, and their applicability to various specific uses. Many of these distinctions are merely local, or are dependent on comparatively slight peculiarities, but the general classification into *caking* or *coking coals*, on the one hand, and *non-caking coals*, on the other, is both definite and of great practical importance. Between these two extremes are numerous sub-varieties, which have, in different localities, received names indicative of their greater or less similarity to one or other of the types.

When caking coals are strongly heated they become partially fused, and, when in a pasty state, swell into a spongy mass, giving off bubbles of gas, which, as it escapes, burns with a bright flame. Coals of this description, when reduced to powder and strongly heated in a covered vessel, agglomerate into a mass of coherent coke. This property of caking varies in degree, in different coals, from slight agglutination to almost complete fusion.

The caking of coal does not, however, take place at a temperature below that at which its decomposition is effected, and consequently it cannot be regarded as the result of a mere fusion of its particles. It is, on the contrary, caused by the action of heat on its constituents giving rise to the formation of coal-tar, which, becoming subsequently charred, cements the whole into a solid and frequently sonorous mass. Even the powder of charcoal or anthracite, if mixed with a small quantity of pitch or coal-tar and strongly heated in a closed crucible, will afford a perfectly solid coke, which, when struck, has an almost metallic ring.

Non-caking coals are those which do not, in burning, cake or sinter together in any sensible degree, and of which the particles, when strongly heated in a closed vessel, do not unite to form a solid coherent coke. A fire supplied with coals of this description remains open, allowing the air to pass freely through it; whereas many varieties of caking coal cannot, without an admixture of a non-caking coal, be employed for metallurgical purposes on account of the stoppage of the air-passages, caused by its agglomeration into a more or less compact and impervious mass.

It would appear from the researches of Professor Stein of Dresden on the coals of Saxony, and from the investigations of Dr. Percy on those of this country, that the property of caking is dependent rather on the proximate constitution of a coal than upon its ultimate composition, and that a caking and a non-caking coal may have the same elementary composition. This subject is, however, worthy of further investigation.

It has been asserted on good authority that certain Welsh coals lose their property of caking after a few days' exposure to the air, and M. de Marsilly states that strongly caking coal, which affords an excellent coke when fresh from the pit, yields an imperfect coke after exposure to the atmosphere for six months. The more important conclusions arrived at by M. de Marsilly may be summarized as follows:—Coal suffers a less loss of weight by desiccation *in vacuo* than by exposure to a temperature of 100° C. It begins to give off gas at 50° C., but its evolution is not very sensible below 100° C. The amount of gas evolved goes on increasing up to 330° C., when the decomposition of the coal, properly so called, probably commences; a liquid product having the odour of benzene is distilled off at the same time. The loss of weight experienced by coal at 300° C. ranges from 1 to 2 per cent.

The caking of coal is also, to a considerable extent, influenced by the way in which it is treated, since, in some cases, a coal which, if heated in the usual way, is practically non-caking, will, when rapidly exposed in a close vessel to a very high temperature, yield a firm coherent coke. The amount of moisture in a coal has likewise a certain influence on its property of caking, and when a large quantity of inorganic matter is present, it is not without effect in diminishing this property, although Stein has found that a coal containing nearly 22 per cent. of ash may still be capable of caking.

Strongly caking coal, from becoming agglomerated on the grate, and thus preventing the free passage of air, is not often adapted for metallurgical purposes, except in the form of coke, and a fuel containing a large amount of a fusible ash may be equally objectionable, from the choking of the grate by the formation of clinkers. If, therefore, a coal possesses, in a high degree, the property of caking, or yields a large proportion of fusible ash, it is usually mixed either with a non-caking variety, or with a coal of which the ash has a tendency to prevent the formation of clinkers, when mixed with the more fusible ashes of the other.

The following table, extracted from the 'Third Official Report on Coals suited to the Steam Navy,' gives the percentage composition of several varieties of British coal, together with their specific gravities and the amount of ash and coke yielded by each.

The results obtained by Regnault and Karsten from the analyses of specimens of various foreign coals are given on page 47.

### Composition of Various British Coals.

| | Locality or Name of Coal. | Specific Gravity. | C. | H. | N. | S. | O. | Ash. | Percentage of Coke left. |
|---|---|---|---|---|---|---|---|---|---|
| Welsh Coals. | Aberaman Merthyr | 1·305 | 90·94 | 4·28 | 1·21 | 1·18 | 0·94 | 1·45 | 85·0 |
| | Ebbw Vale | 1·275 | 89·78 | 5·15 | 2·16 | 1·02 | 0·39 | 1·50 | 77·5 |
| | Thomas's Merthyr | 1·30 | 90·12 | 4·33 | 1·00 | 0·85 | 2·02 | 1·68 | 86·53 |
| | Duffryn | 1·326 | 88·26 | 4·66 | 1·45 | 1·77 | 0·60 | 3·26 | 84·3 |
| | Nixon's Merthyr | 1·31 | 90·27 | 4·12 | 0·63 | 1·20 | 2·53 | 1·21 | 79·11 |
| | Binea | 1·304 | 88·66 | 4·63 | 1·43 | 0·33 | 1·03 | 3·96 | 88·10 |
| | Bedwas | 1·32 | 80·61 | 6·01 | 1·44 | 3·50 | 1·50 | 6·94 | 71·7 |
| | Hill's Plymouth Works | 1·35 | 88·49 | 4·00 | 0·46 | 0·84 | 3·82 | 2·39 | 82·25 |
| | Aberdare Co.'s Merthyr | 1·31 | 88·28 | 4·24 | 1·66 | 0·91 | 1·65 | 3·26 | 85·83 |
| | Gadly Nine-feet Seam | 1·33 | 86·18 | 4·31 | 1·09 | 0·87 | 2·21 | 5·34 | 86·54 |
| | Resolven | 1·32 | 79·33 | 4·75 | 1·38 | 5·07 | included in Ash. | 9·47 | 83·9 |
| Newcastle. | Willington | ... | 86·81 | 4·96 | 1·05 | 0·88 | 5·22 | 1·08 | 72·19 |
| | Andrew's House, Tanfield | 1·26 | 85·58 | 5·31 | 1·26 | 1·32 | 4·39 | 2·14 | 65·13 |
| | Bowden Close | ... | 84·92 | 4·53 | 0·96 | 0·65 | 6·66 | 2·28 | 69·69 |
| | Haswell Wallsend | 1·286 | 83·47 | 6·68 | 1·42 | 0·06 | 8·17 | 0·20 | 62·70 |
| | Newcastle Hartley | 1·29 | 81·81 | 5·50 | 1·28 | 1·69 | 2·58 | 7·14 | 64·61 |
| | Hedley's Hartley | 1·31 | 80·26 | 5·28 | 1·16 | 1·78 | 2·40 | 9·12 | 72·31 |
| | Bates' West Hartley | 1·25 | 80·61 | 5·26 | 1·52 | 1·85 | 6·51 | 4·25 | ... |
| | West Hartley Main | 1·264 | 81·85 | 5·29 | 1·69 | 1·13 | 7·53 | 2·51 | 59·20 |
| | Buddle's West Hartley | 1·23 | 80·75 | 5·04 | 1·46 | 1·04 | 7·86 | 3·85 | ... |
| | Hastings' Hartley | 1·25 | 82·24 | 5·42 | 1·61 | 1·35 | 6·44 | 2·94 | 35·60 |
| Derbyshire. | Earl Fitzwilliam's Elsecar | 1·296 | 81·93 | 4·85 | 1·27 | 0·91 | 8·58 | 2·46 | 61·6 |
| | Hayland & Co.'s Elsecar | 1·317 | 80·05 | 4·93 | 1·24 | 1·06 | 8·99 | 3·73 | 62·5 |
| | Earl Fitzwilliam's Park Gate | 1·311 | 80·07 | 4·92 | 2·15 | 1·11 | 9·95 | 1·80 | 61·7 |
| | Butterly Co.'s Portland | 1·301 | 80·41 | 4·65 | 1·59 | 0·86 | 11·26 | 1·23 | 60·9 |
| | Butterly Co.'s Langley | 1·264 | 77·97 | 5·58 | 0·80 | 1·14 | 9·86 | 4·65 | 54·9 |
| | Staveley | 1·27 | 79·85 | 4·84 | 1·23 | 0·72 | 10·96 | 2·40 | 57·86 |
| | Loscoe Soft | 1·285 | 77·49 | 4·86 | 1·64 | 1·30 | 12·41 | 2·30 | 52·8 |
| Lancashire. | Ince Hall Co.'s Arley | 1·272 | 82·61 | 5·86 | 1·76 | 0·80 | 7·44 | 1·53 | 64·0 |
| | Haydock Little Delf | 1·257 | 79·71 | 5·16 | 0·54 | 0·52 | 10·65 | 3·42 | 58·1 |
| | Balcarres Arley | 1·26 | 83·54 | 5·24 | 0·98 | 1·05 | 5·87 | 3·32 | 62·89 |
| | Blackley Hurst | 1·26 | 82·01 | 5·55 | 1·68 | 1·43 | 5·28 | 4·05 | 57·84 |
| | Ince Hall Pemberton Yard | 1·348 | 80·78 | 6·23 | 1·30 | 1·82 | 7·53 | 2·34 | 60·6 |
| | Haydock Rushy Park | 1·323 | 77·65 | 5·53 | 0·50 | 1·73 | 10·91 | 3·68 | 59·4 |
| | Moss Hall Pemberton Four-feet | 1·258 | 75·53 | 4·82 | 2·05 | 3·04 | 7·98 | 6·58 | 55·7 |
| Scotch. | Elgin Wallsend | 1·20 | 76·09 | 5·22 | 1·41 | 1·53 | 5·05 | 10·70 | 58·45 |
| | Wellwood | 1·27 | 81·36 | 6·28 | 1·53 | 1·57 | 6·37 | 2·89 | 59·15 |
| | Dalkeith Coronation Seam | 1·316 | 76·94 | 5·20 | trace | 0·38 | 14·37 | 3·11 | 53·5 |
| | Kilmarnock Skerrington | 1·241 | 79·82 | 5·82 | 0·94 | 0·86 | 11·31 | 1·25 | 49·3 |
| | Fordel Splint | 1·23 | 79·58 | 5·50 | 1·13 | 1·46 | 8·33 | 4·00 | 52·03 |
| | Grangemouth | 1·29 | 79·85 | 5·28 | 1·35 | 1·42 | 8·58 | 3·52 | 56·6 |
| | Eglinton | 1·25 | 80·08 | 6·50 | 1·55 | 1·38 | 8·05 | 2·44 | 54·94 |
| | Dalkeith Jewel Seam | 1·277 | 74·55 | 5·14 | 0·10 | 0·33 | 15·51 | 4·37 | 49·8 |

### Composition of Various Foreign Coals.

| Locality or Name of Coal. | C. | H. | O. & N. | Ash. | Specific Gravity. | Analysts. |
|---|---|---|---|---|---|---|
| Alais, Dép. du Gard . . | 89·27 | 4·85 | 4·47 | 1·41 | 1·322 | Regnault. |
| Rive-de-Gier, Grand Croix . | 87·45 | 5·14 | 5·63 | 1·78 | 1·298 | |
| Flenû from Mons . . . | 84·67 | 5·29 | 7·94 | 2·10 | 1·276 | |
| Decazeville, Dép. Aveyron . | 82·12 | 5·27 | 7·48 | 5·13 | 1·284 | |
| Epinac . . . . . . | 81·12 | 5·10 | 11·25 | 2·53 | 1·353 | |
| Commentry . . . . | 83·72 | 5·29 | 11·75 | 0·24 | 1·319 | |
| Blanzy . . . . . . | 76·48 | 5·23 | 16·01 | 2·28 | 1·362 | |
| Lippe-Schaumburg . . | 89·50 | 4·83 | 4·67 | 1·00 | 1·279 | |
| Céral, Dép. Aveyron . . | 75·38 | 4·74 | 9·02 | 10·86 | 1·294 | |
| Neroi . . . . . . | 63·28 | 4·35 | 13·17 | 19·20 | 1·410 | |
| Saint-Girons . . . . | 72·94 | 5·45 | 17·53 | 4·08 | 1·316 | |
| Sainte-Colombe . . . | 75·41 | 5·59 | 17·91 | 1·09 | 1·305 | |
| Leopoldinengrube, Up. Silesia | 73·88 | 2·76 | 2·47 | 20·89 | ... | Karsten. |
| Königsgrube, „ | 78·39 | 3·21 | 17·77 | 0·61 | 1·285 | |
| Sälzer and Neuak, Westphalia | 88·68 | 3·21 | 8·11 | ... | 1·288 | |
| Hundsnak, „ . | 90·35 | 3·20 | 6·45 | ... | 1·338 | |

The composition of the ashes of a coal is in a great measure influenced by the nature of the rock overlying the seam from which it is extracted, as, besides containing the inorganic elements originally forming part of the plants, by the decomposition of which the coal has been produced, they will also, to a certain degree, consist of various earthy and siliceous materials, deposited in the pores of the coal by the infiltration of water from the strata above.

The analyses of the ashes of seven varieties of British coal afforded the results given in the following table. The alkalies were not estimated.

### Composition of the Ashes of Coals.

| No. | Name of Coal from which the Ash was obtained. | $SiO_2$. | $Al_2O_3$. | $Fe_2O_3$. | CaO. | MgO. | $SO_3$. | $P_2O_5$. | FeS. | Total. |
|---|---|---|---|---|---|---|---|---|---|---|
| 1. | Dowlais, N. Wales | 35·73 | 41·11 | 11·15 | 2·75 | 2·65 | 4·45 | 0·99 | ... | 98·83 |
| 2. | „ „ | 24·18 | 20·82 | 26·00 | 9·38 | 9·74 | 8·37 | 0·21 | 0·38 | 99·08 |
| 3. | „ „ | 37·61 | 38·48 | 14·78 | 2·53 | 2·71 | 0·29 | 2·00 | ... | 98·40 |
| 4. | „ „ | 39·64 | 39·20 | 11·84 | 1·81 | 2·58 | trace | 3·01 | ... | 98·08 |
| 5. | Bedwas „ | 26·87 | 56·95 | | 5·10 | 1·19 | 7·23 | 0·74 | ... | 98·08 |
| 6. | Porthmawr „ | 34·21 | 52·00 | | 6·20 | 0·66 | 4·12 | 0·63 | ... | 97·82 |
| 7. | Fordel splint, Scotch . . | 37·60 | 52·00 | | 3·73 | 1·10 | 4·14 | 0·88 | ... | 99·45 |

Nos. 1–4 analysed by E. Riley; Nos. 5–7 by J. A. Phillips.

Cannel Coal.—This is a compact, jet-like variety of coal, sonorous when struck, breaks with a conchoidal fracture, and does not soil the fingers when handled. It is said to derive its name from the candle-like light it yields when burning, and is known to Scotch miners as "parrot coal," from the crackling, chattering sound it emits when thrown on the fire. Cannel coal occurs in certain districts interstratified with ordinary coal, and often forms, in the Scotch coal-fields, the upper portion of a seam of non-caking coal, or even of a bed of blackband ironstone. It is rich in *disposable* hydrogen, and is used chiefly in the manufacture of gas, for which purpose it is in great demand. The cannel coals of Wigan are mined in the immediate vicinity of that town, but thin out in every direction from Wigan as a centre.

COMPOSITION OF CANNEL COALS.

| Locality. | Specific Gravity. | C. | H. | O. | N. | S. | Ash. | $H_2O$ %. | Analysts. |
|---|---|---|---|---|---|---|---|---|---|
| Wigan . | 1·27 | 80·07 | 5·53 | 8·10 | 2·10 | 1·50 | 2·70 | 1·91 | Vaux. |
| „ . | 1·32 | 84·07 | 5·71 | 7·82 | ... | ... | 2·40 | ... | Regnault. |
| Tyneside | 1·32 | 78·06 | 5·80 | 3·12 | 1·85 | 2·22 | 8·95 | ... | Taylor. |
| Boghead | 1·20 | 65·72 | 9·03 | 4·78 | 0·72 | ... | 19·75 | ... | Stenhouse. |

Dr. Stenhouse obtained the following results as the mean of three analyses of the ash of Boghead cannel:—

| | |
|---|---|
| $SiO_2$ . . . . . | 58·31 |
| $Al_2O_3$ . . . . . | 33·65 |
| $Fe_2O_3$ . . . . . | 7·00 |
| $K_2O$ . . . . . | 0·84 |
| $Na_2O$ . . . . . | 0·41 |
| CaO and $SO_3$ . . . . | traces |
| . | 100·21 |

**ANTHRACITE.**—Anthracite may be regarded as the ultimate product of the conversion of vegetable matter into coal. Its structure is perfectly homogeneous, its fracture conchoidal, and its colour a jet black, with a vitreous lustre, which frequently exhibits a remarkable play of colours.

The results obtained by various chemists by analyses of specimens of this substance are given below.

COMPOSITION OF ANTHRACITES.

| Locality. | C. | H. | O. & N. | Ash. | Specific Gravity. | Analysts. |
|---|---|---|---|---|---|---|
| Pottsville, Pennsylvania. . | 90·45 | 2·43 | 2·45 | 4·67 | 1·462 | Regnault. |
| Rolduc, near Aix-la-Chapelle. | 91·98 | 3·92 | 3·16 | 0·94 | 1·367 | |
| Mire, Braconnière . . . | 91·45 | 4·18 | 2·12 | 2·25 | 1·343 | |
| Sablé, Dép. de la Sarthe . | 87·22 | 2·49 | 3·39 | 6·90 | 1·751 | Jaquelin. |
| Vizille, Dép. de l'Isère . . | 94·09 | 1·85 | 2·85 | 1·90 | 1·730 | |
| Isère . . . . . . | 94·00 | 1·49 | 3·58 | 4·00 | 1·650 | |
| Glamorganshire . . . | 91·44 | 3·84 | 3·58 | 1·52 | 1·375 | Wrightson. |
| Slievardagh, Ireland . . | 80·03 | 2·30 | ... | ... | 1·590 | H. How. |

**OCCLUSION OF GASES IN COAL.**—The composition of the gases enclosed in coal has of late years attracted considerable attention. Investigations relative to this subject have been made in Germany by Dr. E. v. Meyer,[1] who determined the volume and nature of the gas evolved from the coals of various localities, while still more recently Mr. J. W. Thomas[2] has experimented in this country upon different varieties of British coal. The experiments of v. Meyer were conducted by collecting and examining the gases evolved when pieces of coal of the size of nuts were kept heated in water during many hours at 100° C. in such a way as to avoid actual boiling. Mr. Thomas sawed slices, each about six inches in length, out of the middle of large blocks of the coal to be examined, and, without delay, inserted them into glass tubes of suitable diameter; one end having been previously drawn out into a narrow neck, so as to conveniently form a connection with the Sprengel mercurial pump. The other end was then sealed before the blowpipe, at such a distance as to prevent any material increase of temperature. The usual water-joint connection was then made with the Sprengel pump, the air exhausted as quickly as possible, and the last portions of the gases brought over subjected to analysis. When the whole of the air had been thus removed, the tube containing the coal was immersed in a vessel of boiling water, and kept at a temperature of 100° C. for about seven hours, or until the pump ceased to bring over any appreciable quantity of gas. For the purposes of comparison, Mr. Thomas divides the coals experimented upon into three classes, namely, *house coal*, *steam coal*, and *anthracite*.

The gases evolved from these three classes of coal differ materially both in quantity and in composition. House coal, from near the surface, contains little or no marsh-gas, while the percentage of carbonic anhydride and nitrogen is usually very high. Steam coal occludes a larger volume of gas than does house coal, and it contains a higher percentage of marsh-gas. Anthracite encloses by far the largest volume of gas, the composition of which closely resembles that evolved from steam coal.

The following table gives the composition by volume of the gases obtained from the different coals examined:—

| No. of Sample. | Gas evolved by 100 grammes of Coal at 100° C. (212° F.) in *vacuo.* | Composition of Gases. | | | |
|---|---|---|---|---|---|
| | | Carbonic Anhydride. | Oxygen. | Marsh-gas. | Nitrogen. |
| | C.C. | | | | |
| 1. | 55·9 | 36·42 | 0·80 | ... | 62·78 |
| 2. | 61·2 | 16·77 | 2·72 | 0·40 | 80·11 |
| 3. | 55·1 | 5·44 | 1·05 | 63·76 | 29·75 |
| 4. | 24·0 | 22·16 | 6·09 | 2·68 | 69·07 |
| 5. | 39·7 | 9·43 | 2·25 | 31·98 | 56·34 |
| 6. | 194·8 | 5·04 | 0·33 | 87·30 | 7·33 |
| 7. | 250·1 | 13·21 | 0·49 | 81·64 | 4·66 |
| 8. | 218·4 | 5·46 | 0·44 | 84·22 | 9·88 |
| 9. | 147·4 | 18·90 | 1·02 | 67·47 | 12·61 |
| 10. | 375·4 | 9·25 | 0·34 | 86·92 | 3·49 |
| 11. | 149·3 | 11·35 | 0·56 | 73·47 | 14·62 |
| 12. | 215·4 | 5·64 | 0·54 | 82·70 | 11·12 |
| 13. | 555·5 | 2·62 | ... | 93·13 | 4·25 |
| 14. | 600·6 | 14·72 | ... | 84·18 | 1·10 |

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[1] 'Journal für practische Chimie,' 1871, cxii. p. 42; 1872, cxii. p. 144 and p. 407.

[2] 'Jour. Chem. Soc.,' vol. xxviii. 1875, p. 793.

*House Coals.*—1. Plymouth Iron-Works. 2. Plymouth Iron-Works, South Pit. 3. Cwm Clydach. 4. Bettwys Coal. 5. Llantwit.

*Steam Coals.*—6. Bute Merthyr Colliery, 2 ft. 9 in. seam. 7. Navigation Colliery, upper 4 ft. seam. 8. Dunraven Colliery, upper 4 ft. seam. 9. Cyfartha Colliery, upper 4 ft. seam. 10. Bute Merthyr Colliery, 6 ft. seam. 11. Dunraven Colliery, 6 ft. seam. 12. Duffryn Colliery, 9 ft. seam.

*Anthracite*—13. Bonville's Court. 14. Watney's Llanelly.

The present annual production of coal in the United Kingdom is about 165,000,000 tons.

## PREPARATION OF CARBONIZED FUELS.

Since all the various substances used for fuel are of organic origin, it follows that they are more or less prone to decomposition. Chemical combinations are stable within certain limits of temperature only, and when these points are exceeded, a series of compounds is produced by fresh groupings of the elements of which the original substance was composed. When a substance, such as wood, is strongly heated, the arrangement of its elements is broken up, and new compounds are produced, capable of existing at the higher temperature at which they are formed. The nature of these products will in a great measure depend on the degree of heat which has been employed, as those obtained at one temperature will materially differ, both in quantity and composition, from those formed at another.

The results will, moreover, be essentially different, according as air is excluded from or admitted into the apparatus in which the heating takes place. When air is admitted, the products at first formed are immediately subjected to the action of oxygen, which combines with their elements to form new bodies, and combustion is the result. If, on the contrary, decomposition is effected by heat alone in closed vessels, air being excluded, the process is known by the name of *dry distillation*, and affords the means of collecting and studying the various products obtained at more or less elevated temperatures. This operation is of the greatest importance, as affording the means of modifying various fuels, so as to adapt them to the particular circumstances under which they are to be employed. When a piece of wood or coal is strongly heated, its elements so arrange themselves as to give rise to various gaseous compounds, and these, escaping at an elevated temperature, ignite and produce flame. This combustion affords sufficient heat to cause the non-volatile portion of the fuel to combine with the oxygen of the air, which in its turn produces a fresh supply of gas from that portion of the mass with which it is in immediate contact. In this way combustion is supported until the substance is entirely consumed, as the heat evolved by the combustion of the carbon on the outer surface of the mass causes the dry distillation of the inner portions with which it is in contact;

while the gases thus evolved tend to facilitate the union of the carbon of the outer surfaces with the oxygen of the air.

From the large amount of water contained in many natural fuels it is evident that, when burnt, a portion of the heat evolved must be rendered unavailable, as the water present will carry off by its evaporation a portion of the heat produced. The presence of hydrogen and oxygen, virtually in combination, forming part of the fuel will also give rise to the production of water, which can only be volatilized at the expense of a portion of the carbon consumed.

In order, therefore, to obtain a larger amount of combustible matter in a given weight of fuel, it has long been the custom to expel the aqueous and gaseous portions of such as are required to afford an intense heat, before applying them to the uses for which they are intended. This is the object of charring wood, or converting it into charcoal, which operation has more recently been extended to peat, lignite, and coal. In the latter case the process is called coking, and the resulting product is known by the name of coke. By this means, the different kinds of natural fuels are made to afford a series of artificial ones, respectively better suited for particular purposes than the substances from which they are derived. Their economical preparation, therefore, becomes a subject of importance, not only to the metallurgist, but to all who require the aid of elevated temperatures.

## CHARCOAL.

If we ignite a small splinter of wood, and closely examine the way in which it burns when the lighted end is held downwards, two distinct periods will be observed. When the flame has become weak, from volatile combustible products having ceased to be evolved, except in very small quantities, it is observed gradually to die out, and nothing will remain but the feeble glimmering produced by the slow combustion of a portion of the remaining charcoal, which not affording sufficient heat to admit of the combustion of the carbon by the oxygen of the air, soon ceases. If, so soon as the flame has died out, the chip be placed in a close vessel, such as a test-tube stopped by the finger, it will, from want of air, be quickly extinguished, without any of the glimmering before noticed; and if a piece of wood be at once heated in a close vessel, so as to completely char it without first producing ignition, the volatile matters are driven off, and charcoal is produced without loss of carbon from the action of the air. In the ordinary methods of preparing charcoal on a large scale, both these principles are, to some extent, involved; as in this case a portion of the wood is consumed in order to sufficiently raise the temperature to drive off the volatile constituents of that which remains, while the combustible products of distillation are more or less perfectly consumed. Less frequently charring is effected in large ovens or retorts, and in that case the second principle only comes into play.

Whichever of these contrivances be employed, it is essential that time be allowed for the oxygen to combine with hydrogen to form water, without which these gases unite with and render volatile a portion of the carbon, thereby diminishing the amount of charcoal produced. Karsten, who has carefully examined this subject, obtained the following results, from which the advantage of the slow over the quick method of charring becomes apparent.

| Species of Wood employed. | Percentage Amount of Charcoal obtained by Quick Method of Charring. | Percentage Amount of Charcoal obtained by Slow Method of Charring. |
|---|---|---|
| Young Oak | 16·54 | 25·60 |
| Old do. | 15·91 | 25·71 |
| Young Red Beech | 14·87 | 25·87 |
| Old do. | 14·15 | 26·15 |
| Young White Beech | 13·12 | 25·22 |
| Old do. | 13·65 | 26·45 |
| Young Alder | 14·45 | 25·65 |
| Old do. | 15·30 | 25·65 |
| Young Birch | 13·05 | 25·05 |
| Old do. | 12·20 | 24·70 |
| Birch 100 years old | 12·15 | 25·10 |
| Young Deal (*Pinus picea*) | 14·25 | 25·25 |
| Old do. | 14·05 | 25·00 |
| Young Fir (*P. abies*) | 16·22 | 27·72 |
| Old do. | 15·35 | 24·75 |
| Young Pine (*P. sylvestris*) | 15·52 | 26·07 |
| Old do. | 13·75 | 25·95 |
| Limetree | 13·30 | 24·60 |

The best method of ascertaining the quantity of charcoal yielded by various kinds of wood is to enclose a weighed fragment of the wood in a well-covered crucible filled with charcoal-dust, and having placed it in an assay furnace, the heat should be gradually raised to redness, at which temperature it is kept for about half an hour, when the crucible must be withdrawn from the fire, and allowed to cool previously to being opened. Fine sand may be employed instead of charcoal-dust for the purpose of excluding air from the wood. When the crucible has sufficiently cooled, the fragment of charcoal is withdrawn and weighed. This experiment should be repeated at least twice on each variety of wood, for the purpose of avoiding error.

Preparation of Charcoal in Piles.—The charcoal-burner selects for this purpose a dry locality, sheltered on at least one of its sides, either by a hill or by a portion of the uncut forest; since, if the piles were constructed in an exposed situation, it would be difficult to prevent their being so acted on by the wind as to cause an unequal charring of the wood. When a proper situation has been chosen, which, to prevent the expense of carriage, should not be far removed from the place where the wood has been felled, a circular piece of ground of the diameter of the intended pile is marked out. If the soil be sandy and dry, this is done by merely cutting around it a shallow drain for the purpose of carrying off any rain-water which may fall during the process of carbonization; but should there be any reason for suspecting the dryness of the soil, the surface is slightly raised by a covering of stones, logs of wood, or by the smaller branches of trees. The next operation is to cover the surface with charcoal-dust, obtained from a preceding operation, or, in default of this, a strew of leaves is sometimes employed. A long post is now driven into the ground in the centre of the circle, and it should be of such a length that its upper extremity may extend a little above the top of the intended pile. Around this the wood, which has previously been cut into proper lengths, is piled, as shown in fig. 2. Great care is taken to avoid large cavities between the billets, and, for this reason, those situated immediately around the central post should be made by splitting the larger branches. In making the mound, their thinner edges are placed towards the central post.

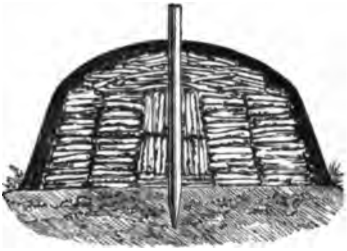

Fig. 2.—Charcoal Pile; vertical section.

The more slantingly the billets are placed against this post, the greater will be the spaces between them; and, therefore, the more nearly perpendicular they can be piled consistently with the stability of the mass and the retention of the external covering, the better will be the subsequent results. It is also evident that, when logs are piled horizontally in concentric circles radiating from the centre, considerable spaces must be produced by the divergence of the outer ends of the billets forming the various rings; and, therefore, a combination of the two methods, as shown in the figure, is frequently adopted. All unavoidable spaces resulting from the crookedness of the branches or their radiation must be carefully filled with small fragments of wood; and when the surface has been thus made even, and the top or cap has been properly rounded by the addition of refuse wood, the pile is provided with its covering. This consists of sods placed on the heap with the grassy side inwards, and beaten all over with a shovel to make them lie closely on its surface. This is again covered with earth, and the whole pressed down for the purpose of giving it solidity. The covering does not, however, extend to the foot of the pile, but is supported at a few inches from the bottom by twigs held in their places by forks, so as to form hoops around the lower part of the pile. This open part at the base of the mound is for the purpose of allowing the escape of the aqueous vapours generated during the first stage of the operation, as no opening is allowed at the upper part of the pile, since it would tend to cause a draught and consume a portion of the wood to be charred.

The dimensions of the pile depend on circumstances incident to the neighbourhood in which the charring takes place, but should in no case be so considerable as not to admit of easy regulation of the temperature. Heaps of only 10 feet in diameter are often met with, but these are, generally speaking, inconveniently small, and piles of from 30 to 40 feet across the base are, therefore, sometimes preferred.

In arranging the billets around the central stake, care is taken to leave at the bottom a small channel extending from it to the exterior of the pile, and by means of this the fire is communicated when it is finished and the external covering has been well pressed down. Sometimes, instead of leaving this opening, the central stake is replaced by three pieces of cleft wood so arranged and tied together with bands of green branches as to form a kind of chimney by which fire may be communicated; and in this case a horizontal passage becomes unnecessary.

When the pile is completed and the covering securely packed down, fire is communicated to the centre of the mass, either by throwing lighted charcoal down the vertical chimney, or by introducing it through the horizontal gallery. For the purpose of facilitating ignition, the wood placed immediately around the stake consists of half-burnt charcoal resulting from a preceding operation, which is picked out for that purpose as being more combustible than ordinary wood, of which the other parts of the stack are composed. When the heap has been ignited, the hole by which the fire has been introduced is closed with turf and earth; and the first period, during which the moisture is expelled from the wood, commences.

At this stage great attention is necessary to prevent the explosion of the heap, from the ignition of a mixture of atmospheric air with the inflammable gases produced by the dry distillation of wood. During the whole of the sweating period, large quantities of yellowish-grey smoke escape, particularly from the uncovered space at the base of the mound, and the interior of the covering becomes moist from the condensation of aqueous vapour expelled from the wood by the action of heat. When the colour of the smoke issuing from the mound has been observed to change to a light grey, without any of the yellow tint before mentioned, the burner closes the open annular space at the base, and the charring period commences.

The covering of sods now requires to be repaired, as the central stake and the dry wood in its vicinity will have become partially consumed, and have caused a sinking of the top or cap of the pile. The upper part of the covering is, therefore, rapidly removed, the charred wood forced down by means of a long pole into a compact mass, and the cavity thus made immediately filled with fresh logs. The covering is now replaced as quickly as possible, and any crevices which may have occurred in it from the sinking of any part of the mound are stopped without delay, as they would otherwise, by admitting atmospheric air, cause the combustion of a portion of the wood. The pile is now left to itself for some days, except that small holes are from time to time made in the covering, from four to five feet above the ground.

This is done chiefly for the purpose of allowing the escape of tarry vapours and not to admit the requisite amount of air, which enters mainly by openings made round the base of the pile.

The dimensions of the pile have, at this stage, become considerably reduced, and care must be taken to observe whether it has equally diminished in all its parts, or if some portions of its surface have sunk, whilst others are in their original condition, thereby giving an irregularity of outline.

If such be the case, the charring has been badly conducted. This may to a certain degree be obviated by covering the sunken or more perfectly carbonized parts with an additional layer of sods, and by means of an aperture made in the raised portion the draught may be increased in that direction.

Towards the end of the process, when the wood in the interior of the pile has become perfectly carbonized, it will be found necessary to adopt means to effect the charring of those portions which are in immediate contact with the movable covering. In this direction the wood is so cooled by radiation and by the condensation of aqueous vapours as to escape carbonization; and the workmen, therefore, accelerate the draught in this portion of the heap by making a second series of holes in the covering, parallel with those which have before been described, but at greater distances from one another. These are allowed to remain open until the smoke that issues from them is seen, by its blue colour, to be free from watery vapour; and, when this period has arrived, they are closed, in order to give place to others made at a short distance below them. Holes for this purpose are never made in the crown of the pile, as the draught is naturally in that direction; but in very large mounds, three or even four successive series of openings are not unfrequently made at different heights above the surface of the ground. The time necessary for the operation chiefly depends on the size of the pile. Small piles are generally carbonized in from six to fourteen days; but if the diameter be more than thirty feet, a month will be required.

If at the termination of the process the covering were removed, and the heap broken up while still hot, the access of air would cause the charcoal to ignite, and the whole would be consumed. On the contrary, if the covering were allowed to remain undisturbed until the mass had cooled down, so as to admit of its being removed without danger, much time would be lost. The charcoal is therefore withdrawn in small quantities, and with suitable precautions. In order to do this, the burner lays bare a space of two or three feet at the bottom of the pile, and, with an iron crook fitted to a wooden handle, withdraws, one by one, the logs of charcoal. These, which are red hot when drawn out, are extinguished either with water or by being buried in sand or in damp charcoal-dust; and as soon as the air begins to act too strongly on the exposed part, the opening is closed, and another made in a different part of the pile. This operation, which is repeated until the whole has been removed and extinguished, is best performed at night, as the slightest spark is then visible, and the chance of loss from the ignition of the charcoal thereby reduced.

In some parts of Lower Austria another arrangement is employed for the preparation of charcoal. This process is said to yield charcoal of a better quality than that obtained by the ordinary method. For this purpose the logs are laid together in the form of a wedge, as shown in fig. 3, of which the breadth is regulated by the length of the logs. Its length varies from twenty to thirty feet. The thick end, which is farthest from that at which the fire is communicated, is from seven to nine feet in height, whilst the thin end is only about three feet above the level of the soil. In the erection of a heap of this kind, the burner commences by driving stakes, *a*, around the parallelogram in which the logs are to be placed. These must project from the surface so as to be of the same height as the pile at the points at which they are respectively driven. Their outline, therefore, in every respect corresponds with the form of the pile itself.

These stakes are so placed as to leave a space all round the wedge-shaped heap of logs piled within the enclosure. The billets are usually eight feet in length; and therefore in order to allow a space of six inches between their ends and the sides of the enclosure, the latter is made about nine feet in width. The opening thus left between the ends of the wood and the stakes is for the purpose of receiving the covering, which, on account of the perpendicular sides, could not otherwise be kept in its place.

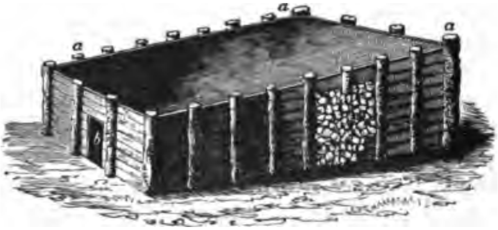

**Fig. 3.—Charcoal-burning; rectangular heap.**

Slabs or pieces of cleft wood are now applied against the inside of the posts, and wet charcoal-dust or *breeze* is stamped down between them and the logs until the interval is entirely filled. When this has been done, the roof receives a covering of twigs, leaves, and, lastly, charcoal-dust and earth, which is moistened with water and well beaten down. In each of the longer sides of the heap a series of holes is made in the boarding, but these do not penetrate the charcoal coating; and in the lower end a larger one, *b*, is left for the purpose of igniting the logs, which is effected by first filling the aperture with shavings or dry wood, and then throwing some red-hot charcoal between the lining and the pile of wood.

When the fire has fairly taken hold of the wood, this aperture is securely closed, and other holes, about three or four inches in diameter, are made at a height of about fifteen inches from the ground, in the same end of the heap. As soon as the smoke issuing from these assumes a clear blue colour, they are stopped, and others made higher up in the pile, which in their turn are closed as soon as the fire has sufficiently advanced towards them. By this arrangement the wood in front of the pile is undergoing the charring process, while that placed behind is merely losing its more volatile constituents. When the operation is drawing to a close, it is found necessary to open a series of holes in the sides of the heap just above the level of the soil, as by this means the charring of the lower logs which constitute the bottom of the pile is effected. These, from their proximity to the ground, and the dampness deposited by the sweating of the wood, would otherwise remain as imperfectly charred billets; and therefore, to prevent this, a series of holes is (when the heap is first constructed) cut in the planking forming the sides. During the early part of the operation these are closed by the charcoal-dust, which is closely packed between the boarding and the wood to be charred. But when the fire is required to descend to the bottom of the heap, the draught is made to pass in that direction by removing the damp charcoal-dust from before the apertures left in the planking. As soon as the logs in front of the pile have become perfectly charred they are removed, and being thus withdrawn from the action of heat shortly after the operation is finished, they not only yield a larger amount of charcoal, but that produced is of better quality than is afforded by the ordinary process.

In the operation of charcoal-burning the dry distillation of the greater portion of the wood is effected by the combustion of a certain quantity, which may be considered as the fuel necessary to produce the required heat. In order to conduct the operation with the greatest possible economy, no more air should be admitted than is absolutely necessary for the combustion of this amount of fuel, as any further supply will cause more to be consumed than is required for the dry distillation of the mass.

The success of these processes is also much influenced by the fact that the smoke and vapours, contrary to their natural course, are made to take a downward direction, which not only affords the workman the opportunity of attentively watching the changes which are taking place, but also gives him time for taking steps to prevent the access of too large an amount of air. Ordinary wood loses from twenty to twenty-five per cent. of its bulk during the process of charring; and, consequently, the dimensions of the charcoal produced from a given quantity of wood are much less than those of the original pile before the application of fire. This diminution naturally tends to produce cavities between the fuel and its covering, which, if formed, would become accessible to air, and thereby cause a useless consumption of wood. In the case, however, of the movable coverings employed for this purpose, this inconvenience can seldom occur, since, in proportion as the size of the heap becomes less through shrinking, the covering will sink with it, and is, therefore, more effective in excluding the air than any fixed roofing which could be substituted for it.

The loss arising from the combustion of a portion of the charcoal is also diminished by the way in which the charring is conducted. The piles are always first lighted at the bottom; and, consequently, the lower portions will have become charred before those parts which are in immediate contact with the covering are much affected by heat. In this way the charcoal which has arrived at the last stage of carbonization will, by the heat which it evolves, cause the dry distillation of the wood immediately in its vicinity, which, from the combustible gases evolved and the burning of a portion of the wood, protects, by a zone greedy for oxygen, the portion already charred from further action of the air.

All these methods of charcoal-burning have, however, the disadvantage of allowing the volatile constituents of the wood to escape, and these are more or less valuable according to the locality in which they are produced. Various plans have been proposed to prevent this loss. For this purpose it has been suggested that the covering of the pile should consist of hurdles covered with clay, into which pipes for carrying off the volatile products could be fitted. Others have proposed that, instead of covering the heap with earth or sand, slaked lime should be employed, so as to combine with, and thereby fix, the pyroligneous acid. Both these plans have, however, been found to fail, as the first destroys the flexibility of the covering, and the second retains but a small portion of the acid produced.

If the method of carbonization in long piles be resorted to, instead of those more usually employed, the gaseous and liquid products of distillation may be collected by an iron pipe placed in the higher end of the heap, which, being connected with a worm-tub filled with water, will discharge any products, which may be condensed, in the liquid form. In cases where the site of the pile can, from the facility of carriage afforded by a stream of water or otherwise, remain stationary, the heap may be built over a funnel-shaped pit lined with clay, which having at its lower part an inclined channel, either of iron or clay, conveys a portion of the impure acid and tarry matters into a reservoir, where they gradually accumulate. The time required for effecting the complete charring of a heap or mound usually varies according to its size, the dryness of the wood, the state of the weather, and other circumstances. Charcoal-burning in large rectangular piles is extensively practised in some parts of Sweden, where the base is sometimes a square of about 20 feet, but is more frequently a rectangle of about 20 by 25 feet. In China the carbonization of wood is conducted in pits, provided with a chimney communicating with the bottom.

After being filled with wood, the top is closed by a covering of earth, and the necessary air is admitted through a channel opposite the chimney, which communicates with the pit half-way between its top and bottom. Five days after lighting, the smoke begins to give indications of the completion of the process, and, when it has become nearly transparent, the pit and chimney are both hermetically sealed. At the expiration of five or six days from the stopping of the chimney, the charcoal has sufficiently cooled to admit of the pit being uncovered for the purpose of its removal. "Stockholm tar" is made in a somewhat similar manner in Sweden and Finland from the root wood of pine trees.

MANUFACTURE OF CHARCOAL IN KILNS.—The kilns used for this purpose in the United States of America are of three principal forms, namely, rectangular, cylindrical, and conical.[1] The first of these is a rectangular chamber with an arched roof from 40 to 50 feet in length, 12 to 18 feet in width, and about the same height to the spring of the arch, which has a rise of about 5 feet to its crown. The capacity varies from 55 to 90 cords of wood, and kilns of the latter size will yield, under favourable circumstances, about 4000 bushels of charcoal.

In the end walls are two doors, each about 5 feet square, the upper ones serving for introducing the wood and the lower for discharging the charcoal. Three series of vent-holes, about 1 foot apart vertically and 16 inches horizontally, are left in the lower part of the walls for the introduction of the necessary air and to allow of the escape of the various products of combustion and distillation. The walls, which are 15 inches in thickness, are strengthened at intervals by brick buttresses, and by bracings and tie-rods.

The wood with which the kiln is charged is cut into 4-feet lengths, piled flat, and as closely as possible; four men are required to charge a 45-cord kiln in one day. When the kiln has been fired, the doors are closed and carefully luted with mortar, which is also employed for closing the vents as soon as the condensable products of distillation have escaped.

The kiln may be lighted by a chimney left in the centre of the charge, or by a channel running longitudinally through the middle, opening at the two lower doors at the ends. The operation is easily conducted provided the walls are tight. The smoke which first issues is yellow, but becomes blue about the fifth or sixth day, when the vents must be hermetically closed. From four to eight days are required for cooling, which is sometimes accelerated by the use of water. This, however, to some extent deteriorates the quality of the charcoal. The whole charge can be drawn by four men in one day, but from 6 to 10 per cent. of imperfectly charred wood is found upon the floor. These *brands* may either be returned with the next charge or be employed as ordinary fuel. The rectangular kiln is, however, going out of use on account of the difficulty of keeping it air-tight when of large size, while small ones cannot be worked economically.

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[1] 'On the Manufacture of Charcoal in Kilns,' by T. Egleston. Trans. American Institute of Mining Engineers, vol. viii. p. 373.

The cylindrical kiln is a chamber of 28 feet in diameter and 12 feet in height, which is sometimes very slightly coned. This is covered by a dome of about 6½ feet rise, and is bound with broad wrought-iron rings.

There are two principal openings—a circular one, about 5 feet in diameter in the roof, and a rectangular one upon the ground level. A large portion of the charge is introduced by the lower opening, while the higher one is used for filling the upper parts of the kiln. The whole of the charcoal is removed through the lower opening. The filling is commenced by placing pieces of wood about 6 inches in diameter radially on the floor, leaving interstices between them as air-channels. Above these are placed about 2 feet of dry wood and brands, keeping a central space about 4 feet in diameter, which is carried to the top of the kiln to serve as a chimney. The wood, which is placed horizontally, is packed as closely as possible, and when no more can be put in through the lower door, it is closed, and the charging continued from above. Shavings and brushwood are then placed in the central chimney, while the horizontal and radial air-channels have been partially filled with the same material during the charging of the kiln. From four to five men can charge a 45-cord kiln in one day.

The firing is effected by a long torch introduced through the bottom door, by which the inflammable materials in the radial air-channels are ignited. Six, seven, or even ten days, according to the size of the kiln, are required for charring, and from four to six days, after closing all the vents, are necessary for cooling. The charcoal is discharged in one day.

The objection of want of stability made with regard to rectangular kilns applies with even greater force to cylindrical ones, and they are consequently being given up in favour of a conical form, which has usually a smaller capacity than either of the two preceding varieties. The usual dimensions are 20 to 25 feet in diameter at the base, with a height of from 20 to 25 feet, taking from 25 to 40 cords of wood, and having the walls so battered as to require no kind of bracing. These kilns, in some cases, form a single truncated cone; in others they are formed of a combination of two cones, of which the flatter forms a roof to the steeper one; while, in a third variety, the walls are coned to a height of about 6 feet only, the remaining 14 feet being curved to the arc of a circle. The doors are arranged and the charging and discharging conducted in a manner generally similar to that described for the cylindrical kiln.

A kiln containing 35 cords of wood requires one day for charging, three days for cooling, and a day for discharging. The yield of charcoal in the best-conducted establishments is about 50 bushels per cord, although in the case of hardwood it may sometimes be as high as 60 bushels. The weight of the charcoal produced varies with the nature of the wood used, white pine being 9·8 lbs. and the hardest sugar maple 18·95 lbs. per bushel.

In all cases it is desirable to face the vents in these kilns with cast-iron, since without such a protection the mortar in their immediate vicinity is rapidly acted upon by pyroligneous acid.

When the collection of the volatile constituents of wood becomes a matter of great importance, the charring is conducted in stationary kilns or furnaces, from which they may be conveyed by proper appliances into receivers adapted for the purpose. Such kilns or furnaces are of two kinds. In the first, carbonization is effected, as in the case of the common charcoal pile, at the expense of a certain portion of the wood, which is consumed in order to produce the heat necessary for the distillation and charring of the remainder. In the second variety, the heat necessary for the dry distillation of the wood is not obtained by the combustion of any part of the charge of the kiln, but is, on the contrary, raised by the expenditure of a certain amount of fuel, entirely distinct from that from which charcoal is to be manufactured.

Figs. 4 and 5 represent a section and plan of one of the many kilns belonging to the first class. In this arrangement the air has access into the furnace through the grate, *a*. This kiln is partially filled through the door, *b*, and when the charge has been raised to that level, the remainder of the wood is introduced through the aperture, *c*, left for that purpose in the crown of the dome. When the charging has been completed, the openings, *b* and *c*, are closed by doors or tiles against which earth is thrown for the purpose of excluding the air. The wood is now ignited by kindling a fire in the ash-pit immediately beneath the grate; and when it has become fairly alight, the draught is regulated by a sliding door, *d*.

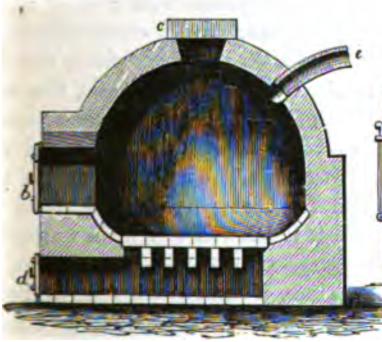

Fig. 4.—Charcoal Kiln; vertical section.

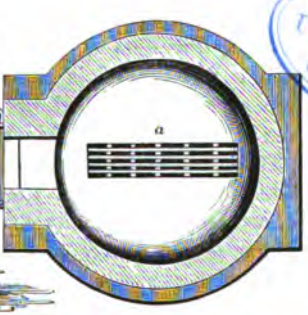

Fig. 5.—Charcoal Kiln; plan.

As soon as the walls of the kiln have attained a sufficient heat to complete the operation, the door, *d*, is closed, and the kiln left to itself, until the whole of the wood which it contains has been converted into charcoal. The volatile ingredients escape through the aperture, *e*, and are more or less completely condensed by passing through a series of tubes surrounded by water.

The grate at the bottom of these kilns, instead of being made of iron bars, is a kind of trellis formed either of open brickwork or of perforated tiles; and when the apertures of these are small, the firing of the charge is effected through the door, *b*. When the operation is terminated, all the apertures by which air could be admitted are completely closed, and the kiln is allowed to cool; the charcoal being finally withdrawn through the door, *b*.

Kilns of different constructions are in various places employed for the production of charcoal, according to this principle. Some of these are made quadrangular, and, in form, nearly resemble the boarded heaps before described. Two openings in the lower end, to which doors are adapted, serve for charging the wood and withdrawing the charcoal when made, while the requisite supply of air is obtained through apertures in the walls. The first row of holes is made on the same level as the ground on which the kiln stands; the second at eighteen inches from the floor; and the third at a distance of about three feet from the first.

When the wood is placed in the kiln, a channel is constructed in the direction of its longer axis, which corresponds with a door at the lower end of the brickwork. By this opening fire is communicated to the charge, and as soon as ignition of the mass has taken place, the door by which it was lighted is closed. As the process proceeds, the apertures are successively closed by plugs of clay, and when the operation has been completed, the whole of the exterior surface is covered with clay. The gases and other volatile ingredients escape through a pipe placed in the higher end of the kiln for that purpose, and when the charge has become completely charred, it is allowed to remain untouched until it has grown nearly cold. In large kilns of this description, the cooling often occupies several days, since, if they were opened before being reduced to a proper temperature, the air, on coming in contact with red-hot charcoal, would cause it to be consumed.

Appliances of the second kind, *i.e.*, where the heat necessary for charring the charge is applied from without, are employed chiefly where the charcoal produced is of less value than the tar, pyroligneous acid, naphtha, and other volatile ingredients, and are therefore seldom used by the metallurgist for the preparation of the fuel he requires. Coniferous woods yield the largest amount of tar, and are consequently those most frequently subjected to distillation when the production of that substance is the important consideration. The arrangements by which both the tar, &c., and charcoal may be made available are extremely various, but one of those best adapted for the purpose consists of a cast-iron cylinder, so set in brickwork that the flame and strongly-heated gases escaping from a fireplace below may circulate freely around it. The wood to be charred is placed in the cylinder through an opening which can be hermetically closed. This is done by a luted door, and the vapours arising during the distillation are conveyed by an iron pipe into a worm-tub, where those products which are capable of assuming the liquid form are condensed. When the wood in the retort ceases to give off tarry vapours, the luted door is removed, and the charcoal is withdrawn by rakes. On being withdrawn from the retort, it falls into sheet-iron cases, which, being furnished with closely-fitting covers, prevent the combustion of the red-hot charcoal, and in these it is allowed to cool. By a more recent improvement in the construction of retorts of this description, the non-condensable gases issuing from the wood in process of charring are made to afford a portion of the heat necessary for carrying on the operation. This is done by conducting the vapours, which escape condensation on passing through the worm-tubs, under the bottom of the vessel in which the wood is heated. When this arrangement has been raised to a certain temperature by a fire placed under the retort, and considerable quantities of combustible gases are evolved, they become ignited on coming in contact with the flame from the grate, and thus afford sufficient heat for completing the operation.

The average weight of charcoal produced by the pile or stack process from ordinary wood amounts to about 22 per cent. When the distillation is carried on in close ovens, this quantity is frequently increased to 27 per cent.; but as about 5 per cent. is required for heating the oven or retort, this method in reality affords results very little superior to those obtained from the common charcoal mound.

Mushet, who made a series of experiments on the amount of charcoal yielded by different kinds of wood, obtained the annexed results. This investigation was conducted on a small scale, and the woods, before being charred, were thoroughly dried, and pieces of each sort selected as nearly alike in every respect as possible. One hundred parts of each kind were employed, and respectively yielded the following amounts of charcoal:—

| | | Per cent. | |
|---|---|---|---|
| Lignum Vitæ | afforded | 26·0, | of a greyish colour, resembling coke. |
| Mahogany | „ | 25·4, | tinged with brown, spongy and porous. |
| Laburnum | „ | 24·5, | velvet black, compact, very hard. |
| Chestnut | „ | 23·2, | glossy black, compact, firm. |
| Oak | „ | 22·6, | black, close, very firm. |
| Walnut | „ | 20·6, | dull black, close, firm. |
| Holly | „ | 19·9, | dull black, loose, and bulky. |
| Beech | „ | 19·9, | dull black, spongy, firm. |
| Sycamore | „ | 19·7, | fine black, bulky, moderately firm. |
| Elm | „ | 19·5, | fine black, moderately firm. |
| Norway Pine | „ | 19·2, | shining black, bulky, very soft. |
| Willow | „ | 18·4, | velvet black, bulky, loose, and soft. |
| Ash | „ | 17·9, | shining black, spongy, firm. |
| Birch | „ | 17·4, | velvet black, bulky, firm. |
| Scottish Pine | „ | 16·4, | tinged with brown, moderately firm. |

The specific gravity of charcoal varies with the nature of the wood from which it is manufactured, the denser varieties usually affording the heaviest charcoal, and *vice versâ*. Hassenfratz, who has made careful experiments on this subject, gives the following specific gravities for different kinds of wood-charcoal, interstices included:—Alder, 0·134; birch, 0·203; white beech, 0·183; oak, 0·155; red beech, 0·187; and red fir, 0·176. Knapp, who made his experiments on a large scale, found the following numbers as the weight of a cubic foot of various kinds of wood-charcoal:—Beech-wood charcoal (split wood), 8 to 9 lbs.; charcoal from split oak wood, 7 to 8 lbs.; pine wood, 5·5 to 7 lbs.; of the softer kinds of wood, 4·5 to 5·5 lbs.

The charcoal obtained from wood by the methods above described is not pure carbon mixed with the inorganic constituents of wood, and on being strongly heated in a closed platinum crucible, a considerable loss of weight is experienced through the expulsion of volatile matter.

The analysis of beech-wood charcoal produced by the ordinary mound process afforded Faisst the following results, exclusive of ash:—

| | |
|---|---|
| $H_2O$ | 7·23 |
| C | 88·89 |
| H | 2·42 |
| O and N | 1·46 |
| Ash = 3·02 per cent. | 100·00 |

Charcoal has also the property of absorbing considerable quantities of water from exposure to the air. Allen and Pepys found that by a week's exposure to a moist atmosphere the charcoal of

| | Per cent. | | Per cent. |
|---|---|---|---|
| Lignum Vitæ gained in weight | 9·6 | Beech gained in weight | 16·3 |
| Fir | 13·0 | Oak | 16·5 |
| Box | 14·0 | Mahogany | 18·0 |

Common charcoal when kept in store contains, on an average, about 12 per cent. of hygroscopic water.

In addition to its facility for taking up aqueous vapours, charcoal possesses the property of absorbing large quantities of any gas by which it may be surrounded.

The following table indicates the number of volumes of different gases which one volume of charcoal is capable of absorbing:—

| | | | |
|---|---|---|---|
| $NH_3$ | 90 | CO | 9·42 |
| HCl | 85 | O | 9·25 |
| $SO_2$ | 65 | N | 6·50 |
| $H_2S$ | 55 | H | 1·25 |
| $CO_2$ | 35 | | |

## PEAT-CHARCOAL OR PEAT-COKE.

The charcoal obtained from peat is subject to disadvantages which prevent its general application to metallurgical purposes. In the first place, peat usually contains a large amount of ash, and yields only from 24 to 30 per cent. of charcoal, which must therefore yield a very large proportion of earthy matter when burnt, and is thus rendered useless for many operations to which it might otherwise be advantageously applied. If we suppose, for example, that a variety of peat contains 10 per cent. of incombustible matter, and affords 25 per cent. of coke, it is evident that the coke thus produced would contain no less than 40 per cent. of ash, which would render it unfit for many of the purposes for which, if more free from ash, it might be employed. Besides this, it is found impossible to transport peat-charcoal to any considerable distance from the place where it is prepared, for, being extremely friable, it soon falls to pieces, and is thereby rendered worthless as a fuel. In blast-furnaces, where it has to sustain the weight of the charges above it, this charcoal is found to crumble and choke the blast, and it can therefore be employed only under steam-boilers, in forge fires, or in reverberatory furnaces.

Charring in Heaps.—From the circumstance of peat being generally cut in the form of rectangular bricks, it admits of being piled so as to leave but few interstices between the blocks ; and from being less readily combustible than wood, the heaps in which it is prepared require a less perfect covering, and may be built of smaller size. The heaps commonly employed for making peat-charcoal are from seven to ten feet in diameter, and four feet in height. After a proper situation has been selected for the mound, a post is driven into the ground, and around this the blocks of turf are placed in concentric rings. At the bottom are four channels, made of the thickness of a brick, for the purpose of admitting air into the arrangement. These, which are at right angles to one another, commencing at the central stake, and terminating at the periphery of the circle described by the base of the mound, are subsequently used for regulating the rapidity of the carbonization. When the peat has been properly arranged, the mound is first covered with an inner layer of moss or leaves and by an outer covering of sods or charcoal-dust, which is well packed down to exclude the air. Some dry wood for igniting the mass is then placed at the bottom of the stake, and the fire is communicated through one of the channels before described. By the alternate opening and shutting of these, the combustion is equally effected in all parts of the mound; and as soon as flame begins to appear from the crown, which for a small space around the stake is left uncovered, the opening is closed with turf and earth, and the completion of the charring accomplished by means of holes made all round in the covering, which, commencing near the top, are brought down by six inches at a time until they reach the bottom of the heap. In this process, as in that of charring wood, the progress of the operation is known from the colour of the smoke evolved. Knapp states the produce in charcoal of the mounds examined by him to have been as follows :—

| | Gave per cent. in Weight. | Gave per cent. in Volume. |
|---|---|---|
| Peat, not completely air-dried . . . . . . | 24 | 27 |
| „ air-dried . . . . . . . . | 27 | 32½ |
| „ from Pfungstadt, very dry . . . . . | 30 | 29 |
| „ of good quality, quite dry . . . . | 35½ | 49 |
| „ from the Siegen district, very good . . | 23 | 40 |

The peat is only withdrawn from the mound after being allowed to cool down to a proper temperature, as the use of water for quenching it is objectionable on account of the facility with which the resulting charcoal becomes reduced to powder. The amount of charcoal furnished by peat when the experiments are made on a small scale is usually greater than those above given, as there is then less loss sustained from the crumbling nature of the product.

Charring in Ovens.—Although the product obtained from carbonization in ovens is not much greater than that yielded by the ordinary process, yet the supply of air and the rapidity of charring are more easily regulated, and consequently the operation is more satisfactorily conducted, when ovens are employed.

One of the forms given to these ovens is that used at the manufactory of arms at Oberndorf, in Würtemberg. These ovens have the form of an upright cylinder 9 feet in height, and are closed at the top by a dome, in which an aperture is left for the convenience of charging. This cylinder is 5½ feet in diameter, and has a fire-brick lining 15 inches in thickness. This is surrounded by the same thickness of sand, which is enclosed between the lining and another 15-inch wall, so that the entire thickness of the sand and walls together is 45 inches. Above the floor of the oven are three tiers of air-holes made by inserting pieces of musket-barrels into the wall, which may be closed by stoppers from the outside.

The door for drawing the charcoal is placed on a level with the floor, and is closed by an iron slab, against which sand is thrown for the purpose of excluding the air. On charging the oven, a channel is left through its axis for the convenience of igniting its contents, which is done by throwing lighted charcoal down the chimney thus made. At the commencement of the operation both the charging hole and lower vent-holes are left open, but as soon as the peat becomes white-hot the former is shut by an iron plate and covered with sand. The lower air-holes are at the same time closed, and those which are placed next above them opened. When smoke has ceased to be given off, the whole of the apertures must be closed, and the furnace and its contents are allowed to cool together. The coking process is completed in from thirty to forty-eight hours, but the oven is seldom fit to draw in less than six days after closing the air-holes. For this reason a series of ten ovens is employed, in order to allow of one being charged and another drawn every day.

Instead of effecting the coking of one portion of the peat at the expense of another part consumed in the same furnace to obtain the necessary elevation of temperature, it is sometimes subjected to dry distillation in a kind of retort specially adapted to that purpose. At Crouy-sur-Ourcq, near Meaux, an apparatus of this kind was formerly employed, but we are not aware whether it is still in operation. Fig. 6 is a sectional view of this arrangement; *a* is the cylindrical coking chamber, the walls of which are heated by the flame and hot air passing through the intermediate flue, *b*. This space itself is divided by partitions of fire-tiles into three stages, through the apertures of which the hot air from the fires, *c*, ascends and heats the coking chamber. In order to avoid loss of heat, a cylindrical hollow space, *d*, is left in the outer wall of the kiln, which, being constantly full of stagnant air, tends to prevent the cooling of the apparatus. The peat is introduced through the opening at *f*, which, as soon as the charging is finished, is closed by an iron plate, and thickly covered with ashes or sand. The flue from the fire-place opens above this aperture, and its outlet is provided with a movable iron cover, *g*, in which there is a hole for the escape of the products of combustion. The bottom of the kiln is closed by an iron plate, *h*, which, being attached to the iron rod, *i*, may be withdrawn or replaced at pleasure. When the carbonization is completed, this plate is withdrawn, and the charcoal allowed to fall into the chamber below. The volatile products which are generated during the process are carried off by the pipe, *e*, and conducted into a condensing apparatus.

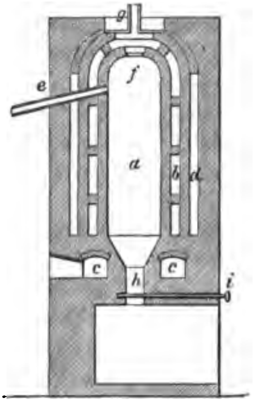

Fig. 6.—Peat-Charcoal Oven; vertical section.

The gases which escape from this are conveyed by metallic pipes into the fire-places, *c*, and being there consumed, yield a portion of the heat necessary for carrying on the operation. The product of charcoal obtained from this furnace is larger than that produced by the methods before described, but from the impact it receives in falling from the retort to the chamber beneath, it frequently becomes much broken, and is thereby rendered to a certain extent less valuable than if obtained in larger masses.

Mr. Vignoles, in the year 1849, patented a process for charring peat by steam superheated to the melting-point of lead or tin. "Peat, in the state of pulp, is thrown in a mass into a cylindrical drum-shaped vessel, divided, if necessary, into compartments, which is caused to revolve with great rapidity on its axis; the requisite velocity being such as shall drive off the water or other fluid from the solid parts of the peat or turf by centrifugal force." The axis of this cylindrical vessel should be vertical, and the cylinder should be from 6 to 10 feet in diameter, and from 1½ to 2½ feet in depth. The external surface is composed of wire-gauze or perforated sheet-metal, of which the apertures should be of such a size as to retain the particles of peat, whilst the water escapes freely through them. When the peat, as obtained from the bog, is not sufficiently pulpy, it may be readily reduced to the state of a nearly homogeneous mud by the operation of edge-stones or a pug-mill. When a sufficient amount of water has been expelled by this process, the peat, in the form of a coherent mass, is removed and moulded into blocks.

These are introduced into large iron cylinders, into which superheated steam is admitted at a pressure of from 45 to 60 lbs. per square inch. The charcoal thus produced is stated to be equal to gas-coke; but this process, although tried on a large scale and at great expense by the inventor, never came into practical use.

The quality of the charcoal obtained will necessarily vary in accordance with the nature of the peat from which it has been prepared. A fibrous peat affords a spongy charcoal, while a more compact variety yields a product of better quality. In all cases, however, the charcoal obtained from peat is friable, and incapable of sustaining handling or moderate pressure without crumbling. This defect would be alone sufficient to render it unsuitable for general metallurgical purposes, since, in the first place, it could not be transported to any considerable distance without great waste, and, in the second, it would be incapable of supporting the pressure to which fuel is subjected in the blast-furnace. Charcoal prepared from peat which has been much reduced in bulk by compression is, to a certain extent, free from this defect; but the preliminary process generally increases the cost to such an extent as to preclude its profitable application. Notwithstanding that peat was, according to Carlowitz, charred in piles as far back as 1712, and peat-charcoal was made in the Harz, as Vogel informs us, in 1735, none of the attempts which have hitherto been made to utilize this fuel have been practically successful on a large scale.

## COKE.

The combustible residue which remains after the volatile constituents of bituminous coal have been eliminated by the action of heat is called coke. In preparing this material, as in the manufacture of charcoal, one of the chief objects is the production of a fuel which shall be more suitable, for certain purposes, than the coal from which it is produced; the elimination of a large portion of the sulphur present in the original fuel is, at the same time, accomplished.

We are without precise information with regard to the date when coke was first employed. It is, however, reasonable to conclude that when, on account of its increasing consumption, wood became scarce and expensive, attempts would be made to extract from pit-coal a material which might be advantageously employed as a substitute for ordinary charcoal. The first experiment would be to subject coal to a process similar to charcoal-burning, and in this way a fuel would be produced of which the practical value must have been quickly appreciated.

Coke was employed in Derbyshire for drying malt about the year 1640. In Plot's "History of Staffordshire" (1686), it is stated that coal was charred in the same manner as wood; that coal so prepared was called "coak," and was capable of producing a heat almost as strong as that afforded by charcoal. Swedenborg, writing in the year 1734, says that in certain districts in England coke was employed in the smelting of iron, but that its use was not brought to perfection. M. de Genssane, in his "Traité de la Fonte des Mines par le Feu du Charbon de Terre" (Paris, 1770), describes and gives drawings of ovens erected at Sulzbach by the Prince of Nassau for the preparation of coke. This arrangement consists of a series of brick muffles or retorts set in a row, in which the coal is distilled by the heat obtained from independent fire-places. The products of distillation were also collected, and are recommended for lubricating cart-axles and for burning in lamps.

In a work published in 1774, M. Jars gives a drawing of nine kilns employed at Newcastle for destroying the sulphur and reducing coal to cinders and "coaks."

Until comparatively recent times coke was most frequently prepared by burning coal in piles or heaps, and even at the present day this method of coking is still practised in certain districts.

Good coke should possess sufficient solidity to enable it to withstand the pressure of a blast-furnace without crushing, and is of little value unless obtained in large prismatic pieces, not liable to crumble and form dust. From the difference observed in the properties and composition of coals, it is evident that all cannot be equally fitted for the manufacture of coke, and it is therefore necessary to select such as are best adapted for the purpose.

The nature of the coke produced from any particular variety of coal is also considerably influenced by the method of its preparation, as well as, in a certain degree, by the nature and amount of the ash existing in the coal from which it is manufactured. Coke which has been made in large quantities is usually better in quality than that manufactured on a more limited scale; as, from the weight of the mass operated on, the product will be more compact than when smaller quantities are used, while, from the high temperature obtained, the volatile ingredients will be more perfectly driven off. When the ash contained in the coal is fusible, it melts and forms a kind of cement for the particles of coke, which is thereby rendered more compact and less liable to be crushed by pressure.

From the circumstance of coal being continuously raised in the same localities, the manufacture of coke is usually carried on by means of stationary arrangements, and, from its being but slightly inflammable and requiring a good draught to effect combustion, its production even in the open air becomes an easy operation.

CARBONIZATION IN HEAPS.—The earliest method of manufacturing coke, and one still employed at the present day, is carbonization in heaps. When this process is resorted to, no external covering of the mound is used; and the coking, which is at first carried on with free access of air, is finally checked by the application of a coating of breeze when coke has already been produced. The coke-heap is always erected on the same station, which soon becomes covered with a sufficient amount of breeze for the purpose for which it is used.

Instead of making round heaps, like the ordinary pile for the production of charcoal, a long, rectangular pile is generally preferred, as by this means a much larger amount of coal can be operated on at one time. The length of these heaps is frequently from 150 to 200 feet; and, in order to erect them, a line is first stretched in the direction of the axis of the heap throughout the whole length of the coke-station. Large pieces of coal are now placed on either side of this, so that by coming together at the top they form a triangular gallery, running the whole length of the intended heap. In making this central channel, it is important the fragments should be so placed as to be nearly upright in relation to the layers of deposition, while the larger surfaces should be at right angles to the axis of the heap. A second series of blocks of coal is then placed on the first, and these are again covered with others similarly arranged, but of gradually decreasing size, until the heap has attained a width of 6 feet on either side of the central channel. The whole is now covered with a coating of smaller coal about 2 feet in thickness, arranged without any regard to its foliation, although care is taken that the larger fragments be placed on the sides of the mound, the top being covered with coal-dust only. In order to ignite the heap, vertical stakes are placed at regular intervals along the central channel; and these, being subsequently withdrawn, leave a series of chimneys into which burning coals are introduced. The fire is thus communicated to the mass in so many points at the same time, that ignition soon becomes general and coking commences throughout its whole extent.

The person in charge of the operation has to prevent the action from advancing too far, and as soon as he observes that thick smoke and flame have ceased to be evolved from any particular part of the heap, and that it begins to get covered with a coating of ash, he prevents further action by covering the flame with damp breeze, which, being closely packed down, prevents the entrance of air, and quickly deadens the fire. As the coking advances this is repeated until the whole heap is covered, when it is allowed to remain two or three days to cool; care being taken to supply it with a thicker covering on the side which is exposed to the wind than on that opposite to it. When the fire has become nearly extinguished the coke is withdrawn, and subsequently cooled by the use of water. This method of making coke, although simple, is far from economical; for inasmuch as the charring is most active on the outer and upper parts of the heap, and gradually proceeds towards the central and lower portions, it follows that the surface is coked before the central parts are reached, which must, however, be fully charred before the air can be excluded. The outside of the pile is therefore burning to waste without the possibility of prevention, while the lower and central portions are comparatively little acted upon.

Coking in Mounds.—A more economical method of making coke than that above described is represented by the annexed woodcut, fig. 7. A chimney of about three feet in diameter at bottom is loosely built with bricks on the coke-hearth or site of the intended mound, BB. The brickwork of this flue has a number of holes, *b*, made by leaving out a brick here and there, and the upper part, which is made of solid work for a short distance near the top, is provided with an iron cover. Around this chimney the coals to be coked are placed in a slightly inclined position; the largest masses being piled nearest the centre of the mound, and the dimensions of the pieces gradually reduced towards the outside. A mound of this kind is generally about twenty-five feet in diameter, and five feet in height; and when completed, its surface, except round the bottom, to the height of about a foot, is covered with a coating of damp coke-dust four or five inches in thickness, which is well packed for the purpose of excluding the air. A shovelful of burning coal is now introduced through the perpendicular chimney, which soon communicates with the mound through the apertures, *b*, and the mass gradually becomes ignited, beginning from the bottom, and from thence spreading in the direction of the covering of breeze. Openings in the foot of the mound, where there is no covering, admit of a certain quantity of air passing through and escaping by the chimney. At the expiration of four or five days the fire will have reached the covering, which becomes red-hot. The top of the chimney is now closed by an iron plate, and the whole surface of the mound, including the uncovered space around the foot of the pile, is at the same time secured by damp tightly compressed coke-dust. After being allowed to remain for two or three days the coke will have sufficiently cooled, and may be drawn and quenched with water.

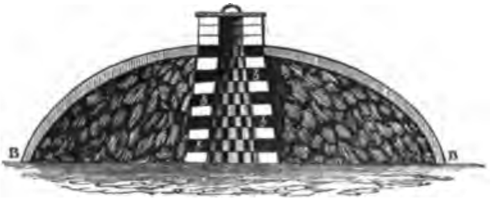

Fig. 7.—Coke Mound; vertical section.

In some localities no covering is applied to these mounds, and when the burning coals are thrown into the central aperture, combustion is carried on by the air, which on all sides enters freely through the crevices occurring between the fragments of coal of which it is composed. When any part of the mound becomes coated with ash, it is immediately covered with damp breeze to protect it from further action, and as soon as the whole surface is thus protected, the iron plate is placed on the top of the chimney and the mound allowed to cool.

By this method a less percentage of coke is produced than is obtained from that now to be described, but it has still some advantages over the ordinary heap.

Coking in Rectangular Kilns.—This method of coking has for many years been practised in Upper Silesia. The kiln consists of a rectangular enclosure (figs. 8, 9, and 10), about 60 feet in length by 15 feet in width, outside measure, which is floored with fire-bricks set on edge, beneath which is a layer of furnace-slag broken small, whereby proper drainage is secured. The inner surface of the walls, which are 5 feet in height, and, laterally, 8 feet apart in the clear, is lined with fire-brick, whilst the outside is constructed either of ordinary brick or rubble-work. In each of the lateral walls, *a*, is a series of openings, *c*, two feet apart, and the same distance from the floor-line, so placed that those on the one side are respectively opposite to those in the opposite wall. From each of the openings, *c*, is a vertical chimney, *d*, ascending to the top of the wall. In order to charge a kiln of this description, one of the openings, *e*, in the end, is closed by brickwork, plastered with clay, while through the opposite one the small coal or slack is wheeled in and spread over the bottom, where it is watered and stamped into a layer reaching to the lower edges of the openings, *c*. Wooden poles, six inches in diameter at the one end and four at the other, are now inserted in the lateral openings on one side, in such a way that their smaller ends rest in the

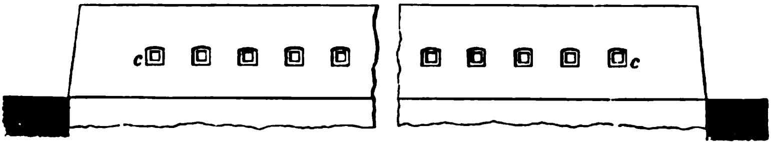

Fig. 8.—Rectangular Kiln; side elevation.

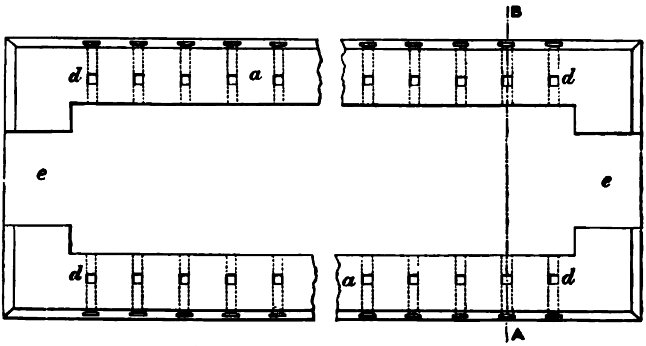

Fig. 9.—Rectangular Kiln; plan.

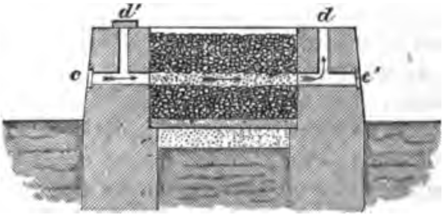

Fig. 10.—Rectangular Kiln; section on the line A B (after filling).

corresponding apertures in the opposite wall. Wetted slack is afterwards thrown upon the pieces of wood and carefully stamped down, and the kiln is then filled up with damped slack, of which every layer, of from six to eight inches in thickness, is consolidated by treading. When the whole of the slack has been introduced, it is covered with a thin layer of coke-dust or clay, and the opening in the end of the kiln, through which the charge has been brought in, is built up with loose bricks and plastered with clay. When this has been done, the pieces of wood are carefully withdrawn in the direction of their larger ends, and in this way a series of channels is formed which is essential to the success of the operation.

Before lighting, all the chimneys on one side are stopped by placing a brick, *d′*, on the top of each (fig. 10), those on the other side being left open. On this side the draught-holes are stopped by bricks, *c′*, the holes on the opposite side being open. The kiln is lighted by means of chips and shavings of readily inflammable wood, inserted in all the unclosed openings, *c*, and a current of air is established, which traverses the channel in the direction indicated by the arrows. Six or eight hours after lighting, the fire will have reached the opposite ends of the channels, when the chimneys on the left, *d′*, and the holes on the right, *c′*, must be opened; at the same time the chimneys on the right, *d*, and the draught-holes on the left, *c*, must be closed. According to the state of the weather and the quarter from which the wind is blowing, the direction of the currents through the coal may be changed every two or three hours, and should the coking be found to proceed irregularly, it may be expedient not to change the direction of the currents in all parts of the kiln at the same time.

The work of the coke-burner consists chiefly in keeping open the transverse channels through the coal, and for that purpose he is provided with a slender iron rod which is somewhat bent at one end. When a channel has once become stopped, considerable difficulty is often experienced in re-opening it; and if several of the neighbouring channels are closed at the same time, the progress of the operation is locally arrested and the results obtained are unsatisfactory. Any cracks which, during the operation, may occur in the covering on the top of the coal must be carefully stopped, and in windy weather the draught through the kiln must be regulated by more or less completely closing the various chimneys and draught-holes. On a judicious regulation of the draught will very much depend the quality as well as the yield of coke obtained.

Under ordinary circumstances, the process will be completed in from eight to ten days. This is indicated by the escape of white flame from the chimneys, and by the resistance experienced on the insertion of an iron rod through the covering of the kiln. The openings are now all carefully closed and plastered with clay, and at the expiration of from two to three days the wall, stopping one of the ends, is taken down and the coke extinguished and removed. By this method of treatment some varieties of Silesian coal are stated to yield 80 per cent. of coke, which is more like charcoal in structure than that made in ovens.

In 1854 Mr. E. Rogers of Abercarn communicated a paper to the Institution of Mechanical Engineers in Birmingham on the manufacture of charcoal and coke by this process, which, as far as it relates to the production of coke, he believed to be new. Kilns, in some cases of not less than 15 feet in width from wall to wall, inside measure, were erected at various establishments in South Wales, and the flues, across the interior of the kilns, were carefully constructed of large blocks of coal, corresponding to the air-holes in the opposite sides. Mr. Rogers was of opinion that kilns 14 feet in width, 90 feet in length, and 7 feet 6 inches in height, would be found more economical than smaller ones. A kiln of these dimensions contains about 150 tons of coal. In South Wales the saving in working expenses was stated to amount to 50 per cent., and the yield of coke to be 75 per cent. of the coal employed. The coke was also stated to possess a greater density and to be of more uniform quality than that usually produced in the district. Opinions on this subject did not, however, agree; and in some establishments at which kilns of this description were erected, they were, after continuous trials, abandoned.

COKING IN OVENS.—The foregoing methods of coke-making are now generally superseded by oven-coking, particularly in localities where, the price of coal being comparatively high, it becomes of importance that as small a proportion as possible should be lost during the operation. The ovens used for the production of coke are for the most part heated by the combustion of a portion of the fuel with which they are charged; and not at the expense of a distinct quantity, specially consumed in a separate fireplace, for the purpose of raising the temperature to the requisite degree.

When, on the contrary, coal is coked in order to obtain the gaseous products of distillation, and the coke is regarded as a secondary product of the manufacture, the operation is carried on in close vessels heated by distinct fires. But, in this case, the coke produced is usually spongy and unfit for many metallurgical purposes.

The ovens employed for the manufacture of coke vary so much in form and dimensions that it would be impossible to give even a brief description of all those in general use; but the principles involved being the same in nearly all cases, it will be sufficient to explain the construction and management of some of those most frequently employed. Fig. 11 represents a range of coke-ovens of the round or "beehive" form. The cavity of this oven is about 9 feet in diameter and 3 feet 6 inches in height, internally lined with fire-bricks well jointed in refractory clay. The form somewhat resembles that of a compressed beehive, and at the top of the dome is a circular aperture or chimney which may be closed by an iron plate. A slightly arched doorway of about 2½ feet square is also left at *b* for the purpose of charging the oven and withdrawing the coke. This opening is strengthened by a heavy cast-iron framing, *c*, built into the brickwork and secured in its place by iron binders. Where large quantities of coke are manufactured, the ovens are placed back to back in double rows, with a series of doors on either side of the long mass of masonry. The charge of each is about three tons of coal, which is introduced either through the door, *b*, or through a charging-hole in the top; and as this is done immediately after the withdrawal of the former charge, and while the oven is still hot, the coal soon begins to give off inflammable gases, which escape through an aperture in the dome. When the charging is finished, the doorway, *b*, is closed by fire-bricks loosely piled up, but the air has still free access into the oven, through openings left in the stopping, which, supplying the necessary oxygen to the gases evolved, they ignite, and the temperature of the oven and its contents is rapidly raised. At the expiration of about three hours, the lower holes in the loose brick wall of the doorway are closed, to prevent the access of too much air, which still finds its way through those above and escapes by the chimney. In some instances, instead of closing the doorway with loose bricks, an iron door, lined with tiles and provided with draught-holes, is employed. These are successively stopped by lumps of clay, and when it is required to exclude the air entirely, it may be done by applying a sheet of iron against the bricks and luting the edges with clay.

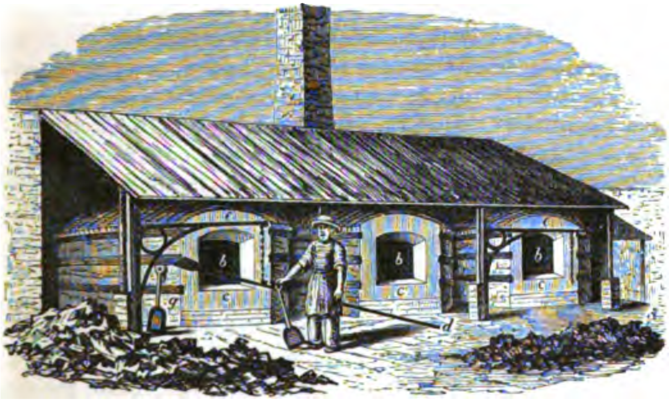

Fig. 11.—Coke-Ovens.

In twenty-four hours after charging, the upper holes are also closed with clay, and the oven is allowed to remain twelve hours with the chimney open, during which time the remainder of the gaseous matter is expelled by the heat and passes off in flame through the opening in the dome. When the flame emitted from this opening ceases, which usually occurs at the expiration of another twelve hours, it is covered with an iron plate made tight by a layer of sand, and the whole allowed to stand for twelve hours, for the purpose of moderating the heat of the oven and its contents.

At the expiration of forty-eight hours from the time of charging, the oven will have sufficiently cooled to admit of drawing, and the door may be opened without causing much loss by the burning of the coke. A large iron shovel, *d*, suspended by a piece of chain to the crane, *e*, is now thrust between the coke and the bottom of the oven, and as the weight on the blade will be supported by the crane, which turns on its axis, a large mass can be taken out with comparatively little exertion on the part of the workman. The swinging of the small iron crane enables the burner to place the heated coke on any part of the paved floor within the circle described by its extreme end from its point of suspension, and as soon, therefore, as it is withdrawn it is strewed thinly on the ground and rapidly cooled with water. When it has been thus extinguished, and has become nearly cold, it is taken up on a grated shovel, *g*, and transported in wheelbarrows either to the place where it is stacked or to the furnace in which it is consumed. The grated shovel is used to separate the breeze from the large coke, as the former falls through the intervals between the bars, so that the latter only can be taken up and placed in the barrows. Iron is used for the barrows, both on account of its being incombustible and also because wooden ones would sooner become destroyed by constant friction against the hard and rough edges of the coke. Fig. 12 represents a ground-plan of an oven of this kind.

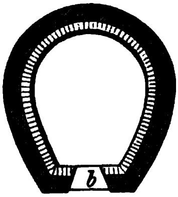

Fig. 12.

Various contrivances have at different times been employed to facilitate the cooling of the ovens after the last stage of coking, but few of these have come into general use, as the increased expense incurred is in most instances more than equivalent to the advantages to be derived from their use. One of the methods of hastening this cooling of the coke is the construction of ovens with air-flues, not only under the refractory lining of the bottom, but also round the sides. During the first stages of the process, all connection between these and the external air is cut off by dampers, but as soon as it is required to cool down the oven for the purpose of drawing, the air is admitted by side openings, and escapes through a set of holes left in the upper part of the brickwork. This causes a draught, and the constant influx of fresh cold air, which, by reducing the temperature of the walls, shortens the usual period allowed for cooling. Where many ovens are employed, and in localities where it is desirable that the smoke produced shall be carried as rapidly as possible from the neighbourhood, a flue is constructed along the whole range, and by this the smoke and fumes issuing from the openings in the crown of the ovens are carried off to a suitable chimney. In this case the flue-holes are usually shut by sliding dampers worked either by a lever or by a pulley and counterpoise.

Some manufacturers are in the habit of cooling the coke in their ovens before it is drawn, which is done by watering it, as soon as the operation is completed, by a jet, under considerable pressure, supplied by a hose. This is said not only to effect a saving by preventing loss by combustion during the time the charge is being withdrawn, but also, by the decomposition of water, to carry off sulphur in the form of sulphuretted hydrogen. Care is of course taken that the oven be not so far cooled as to be unable to inflame the succeeding charge; and, as the whole of the water is converted into vapour before reaching the walls, they are not, it is stated, injured. Coke thus cooled in the oven is said to be brighter and more sonorous than that manufactured in the ordinary way.

The old beehive coke-oven, consisting essentially of a cylindrical chamber reaching the ground level, covered by a dome-shaped roof, although frequently made use of, is not so extensively employed as formerly. A still less expensive form of oven was formerly much used among the collieries of the North of England. This arrangement has the advantage of retaining the heat, and of requiring but little brickwork for its construction. Each oven is a rectangular chamber ten feet deep, twelve feet wide, and ten feet high. The top is arched, and the whole thickness of the side walls, including the internal lining, is two feet. In the centre of the arch an aperture two and a half feet in diameter, lined with a cast-iron ring, is left, whilst another is made on the level of the floor. This opening is sometimes provided with an iron casing with a groove, in which slides the door. The door, which consists of an iron framing filled with brickwork, is suspended by a chain, and is raised or lowered by a lever. In the brickwork of the door several openings are left, which are, however, sometimes dispensed with, and in that case the bricks are loosely placed in the frame, and admit sufficient air through the crevices between them for the combustion of the gases from the coal.

The charge, which reaches the base of the arch, requires to remain from forty-eight to seventy-two hours in the oven before it is in a fit state to be drawn, which is done by an iron hook used specially for that purpose. In all other respects these ovens are managed like those last described; fresh coal being thrown in as soon as the coke is withdrawn, and a certain amount of air admitted during the earlier period of the operation.

Coke produced in ovens is usually denser than that obtained in heaps or mounds; but, on the other hand, it is said to contain a larger amount of sulphur than does coke prepared in the open air. On cooling, good coke splits into prismatic masses, in some degree resembling basaltic columns. Its colour is steel-grey, almost approaching in some instances to silvery whiteness; but the surfaces of many varieties are covered with an iridescence said to be dependent on the presence of sulphur.

Improved Coke-Ovens.—Numerous patents have at various times been taken out which have had for their object the utilization of the heat afforded by the ignited gases which are generally allowed to pass directly into the chimney or into the open air.

*Breckon and Dixon's Ovens.*—The invention of Messrs. Breckon and Dixon consists in constructing coke-ovens with flues for the purpose of conveying the gases, when in a state of combustion, beneath the floor of the ovens.[1] These flues communicate with the interior, and the gases are taken through them before passing into the chimney. The following description will, with the aid of the woodcuts, enable this arrangement to be readily understood.[2] Fig. 13 represents, in elevation, a row of bee-hive coke-ovens provided with a chimney, *k*, through which the gaseous products make their escape.

Fig. 14 is a plan of four coke-ovens, shown partly in section. Fig. 15 is a sectional elevation of two ovens, showing flues leading to the chimney. Fig. 16 is a front elevation, partially in section. These ovens are shown with regulating valves and iron distributors, for which a patent was granted March 29th, 1858, but they are by no means essential to the efficient working of the apparatus. *a* is the door of the coke-oven; *b*, the opening at the top; *c*, the flues which convey atmospheric air, admitted by the regulating valves, *d*, to the distributors *e*; *f* (fig. 16) is the hydrant.

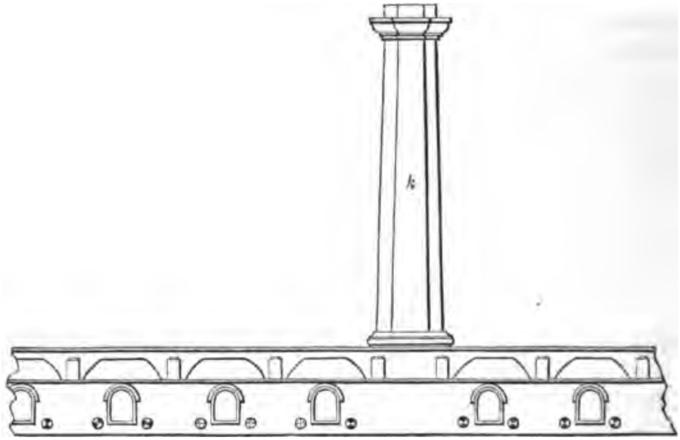

Fig. 13.—Breckon and Dixon's Coke-Ovens; front elevation.

The improvement claimed consists in the application of flues, *g*, constructed under the floor, *h*, of the ovens. These floors are made of fire-tiles, or other suitable material, resting on the division walls of the flues, *g*, which are connected with the interior of the oven by the upright flues, *i*, and the circular openings, *j*, and with the chimney, *k*, through the flues, *l* and *m*. Each orifice, *j*, is in communication with one of the outer flues, *g*, and passes from one to another as indicated by the arrows. When these gases arrive at the two central flues, they ascend the passages, *l*, and enter the horizontal flue, *m*, through which they escape to the chimney. A damper, *n*, placed in each of the flues,

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[1] Specification filed June 9, 1860.

[2] Drawings from 'Transactions of North of England Institute of Mining Engineers,' 1860–61.

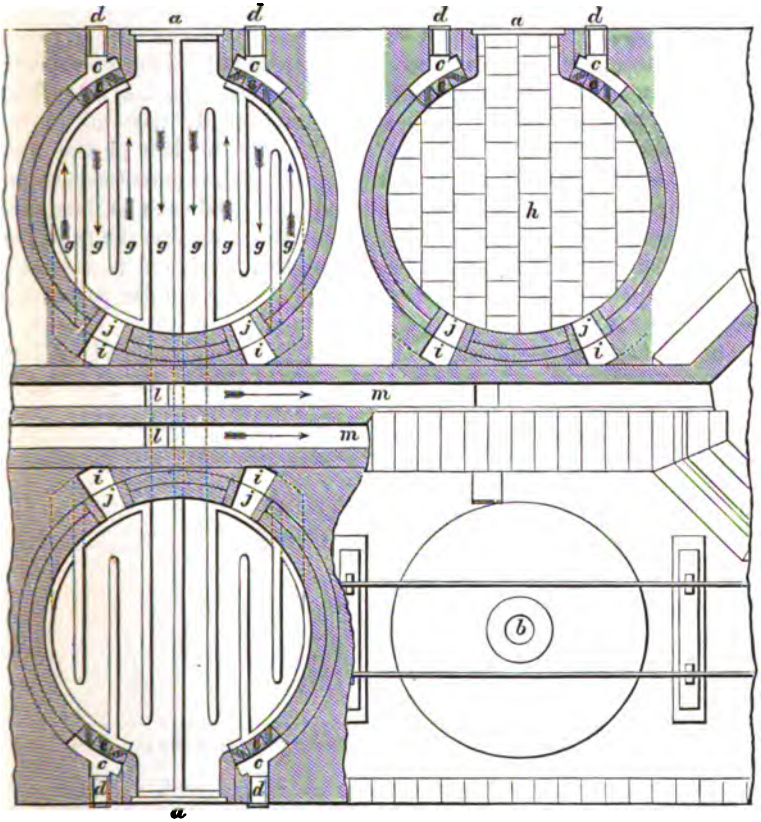

Fig. 14.—Breckon and Dixon's Coke-Ovens; plan of four, partly in section.

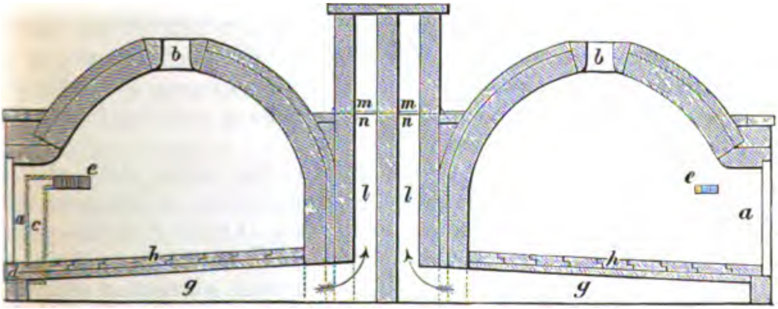

Fig. 15.—Breckon and Dixon's Coke-Ovens; sectional elevation.

*l*, admits of one or more of the ovens being shut off during repairs, or otherwise, without interfering with the action of the others. The coal, with which the oven is charged, may either be dropped through the opening, *b*, or thrown in through the door, *a*.

When the charge becomes ignited, the gaseous products of combustion, escaping through the openings, *j*, descend the flues, *i*, into those under the floor, *g*, where they circulate, and finally pass off to the chimney through the flues, *l* and *m*. In this way the floor of the oven is heated by the ignited gases, which, according to the patentees, enables a given quantity of coal to be converted into good coke in less time than is required in an ordinary oven. They also state that the coke produced is denser and of better quality, and that the yield is increased to the extent of from 10 to 15 per cent. A very similar patent was taken out by Joseph Dunning in May 1853.

It was found at one of the collieries in the neighbourhood of Darlington, after a trial extending over several years, that coal yielding 58 per cent. of coke in ordinary ovens afforded 69 per cent. in those constructed with flues beneath the floor, and that a charge of six tons, which requires seventy-two hours for conversion into good coke in the former, is in the latter well coked in forty-eight hours.

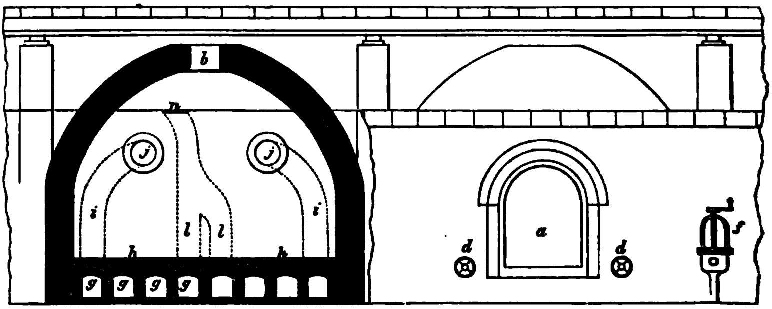

Fig. 16.—Breckon and Dixon's Coke-Ovens; front elevation, partly in section.

*Anchor Oven.*—This rectangular arched oven admits of the entire charge of coke being drawn at once previous to cooling, thus saving a considerable part of the labour required for discharging the ordinary coke-oven, and also any wear and tear to the lining resulting from cooling the ovens by the introduction of water.

In these ovens (fig. 17) the mouth is of the entire width of the chamber, A, and the *anchor*, *d*, *e*, *f*, of wrought-iron, is placed in the oven before it is charged with coal, and is thus, at the end of the operation, embedded in coke. When the oven is to be drawn, a chain is attached to the anchor from a winch fixed in some convenient position, which, when set in motion, draws out the whole charge in one mass upon a paved flooring, where it is cooled by the application of water. In some cases, instead of employing water, a sheet-iron cover is put over the coke, which, around the lower edge, is kept almost air-tight by the application of damp breeze. In this way the atmosphere is so nearly excluded that the coke may be allowed to cool gradually, without experiencing any material loss of weight through waste by burning. The great size of the mouth of the oven, required when this method of drawing coke is resorted to, has been urged as an objection to the use of the process, but this, in practice, is found to be productive of but little inconvenience. The wear and tear of the anchor itself is also an apparent objection, but experience shows that the waste of iron is in reality not great.

The time required for coking in ordinary ovens is usually from forty-eight to seventy-two hours, but when an exceedingly hard coke is required, the operation is sometimes continued during ninety hours. The total cost of manufacture, exclusive of taxes, agency, and redemption of capital, varies from about 1*s.* 4*d.* to 1*s.* 6*d.* per ton.

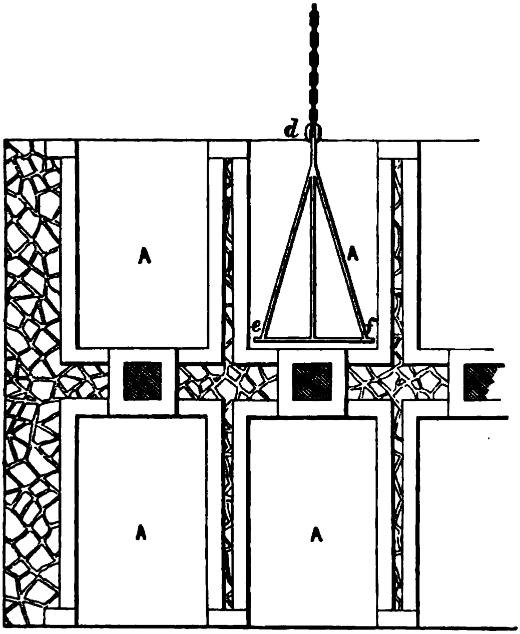

Fig. 17.—Anchor Ovens; ground plan.

COLLECTION OF TAR, &C., FROM COKE-OVENS.—In this country large quantities of tar and ammoniacal liquors are annually furnished by the distillation of coal, from which gas for illumination is obtained, but the demand for such products has of late years become so large that it is now a matter of importance that as small a portion as possible of these substances produced in coke-ovens should escape condensation.

*Pauwels and Dubochet's Oven.*—In 1850 a patent was granted to

Messrs. Pauwels and Dubochet for the manufacture of coke and gas.[1] This invention is described as having for its object, first, the extraction from pit-coal of a gas fit for illumination, and the production at the same time of a coke possessing all the properties requisite for smelting ores and generating steam in locomotives. Second, the regulation, according to circumstances, of the pressure of gas in the passages, so as to render the loss of gas as small as possible. These various results are obtained by means of distinct apparatus; first, an oven or retort, with its extractor, and second, a moderator. The oven or retort is constructed of bricks, cast- and wrought-iron; it is furnished with fire-places for producing the necessary heat by the combustion of coal, coke, or other fuel, and has various flues for the circulation of heat, with a peculiarly constructed heat-magazine; and, lastly, is furnished with divers apparatus, some used as channels for the gaseous products, and others serving, either permanently or at intervals, to isolate the distilling apparatus, properly so called.

The object of the extractor is to protect the apparatus from pressure; it is divided into three distinct parts. The first is composed of three vats full of water, in which as many bells or movable chambers are caused to work up and down by any convenient motive power made to act upon suitable shafts and cranks. The second part of the apparatus consists of two large cylinders, provided with plungers, united together and acting as valves, which constitute, with the first part of the mechanism, an aspirating and forcing apparatus, the action of which is regulated by the third portion of the apparatus, which consists of a large vat in which works a bell or movable chamber, something like a gas-holder. The office of this is to regulate and maintain the equilibrium of pressure.

In order to set an oven in operation, it must first be raised to a high temperature by the application of heat, both to the interior and exterior. This being done, the fires are to be kept up, and as soon as the coal has been introduced into the oven the doors must be closed, leaving open, however, for a few minutes, the orifice of the extraction chimney, to allow aqueous vapours to escape; after which the orifice is closed by means of the cover, and a hydraulic valve is opened at the same time to allow the gas to pass through the pipes, which, as well as the interior of the oven, are protected from pressure by the communication then established between them and the extractor. The gas is thus aspirated or drawn by the extractor and forced into the various apparatus of which gas-works are composed. The moderator must be so regulated that its action shall prevent any variation of pressure on the apparatus and its appendages. Coke-ovens on this principle have been employed at establishments in the neighbourhood of St. Etienne, and at some other places in France. The results obtained are said to have been satisfactory, but the coke produced in such appliances is darker in colour and less dense than that manufactured in ordinary ovens at a higher temperature.

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[1] The system of Pauwels and Dubochet, as well as that patented by Pernolet, are described very nearly in the words of their respective specifications. The author must, therefore not be held responsible for the statements made; they possess, however, considerable interest as showing the direction given to the earlier attempts to render available the various condensable products evolved in coke-making.

*Pernolet's Coke-Oven.*—A patent was granted in 1862 to Richard Archibald Brooman (being a communication from Charles Claude Philibert Nicolas Pernolet) for "Improvements in Coking Ovens, in collecting and utilizing the Products from the Distillation or Carbonization of Coal and other matters producing Coke, and in apparatus employed therein." The invention is described as consisting in the construction, arrangement, and working of coke-ovens in such a manner that the following products are obtained during the manufacture of coke:—First, coke suitable for metallurgical, railway, and other purposes, in greater proportion than when manufactured in the ordinary manner; second, gas suitable for burning and heating; third, tar and different oils obtainable therefrom; and fourth, ammonia and ammoniacal salts.

The apparatus employed is said to be applicable not only to ordinary coal, but also to anthracite, coke-dust, peat, wood, and other combustible materials, of vegetable or mineral origin, whether treated separately or mixed with rich coals, resins, or tars. "Nine chief features are comprised in this invention." They consist—First, in particular arrangements of coking ovens capable of manufacturing coke from all kinds of coal. Second, in the application to these ovens of a continuous exhaust or draught for drawing off and collecting all volatile products resulting from the distillation of combustible matters. Third, in a method of condensing such of those products as are capable of being liquefied, and in the preservation of each of such products so as to utilize them. Fourth, in the application of the gas produced in the ovens, whether to heating the furnaces of the coking ovens themselves or other furnaces, or to the reduction of oxidized ores, or to any other purposes for which carburetted hydrogen can be employed, especially for lighting purposes. Fifth, in pulverizing certain coals which would otherwise produce coke of a bad appearance and ill adapted for metallurgical and railway purposes, whereas, after pulverization of the coal, coke suitable for locomotives and other uses may be produced, and that without washing being necessary. Sixth, in mixing rich coal-tar or resin with poor coal, anthracite, coke-dust, peat, and other combustible materials which do not by themselves produce good coke. Seventh, in the employment of steam thrown on the coke when at its highest temperature for desulphurizing it. Eighth, in the employment of apparatus for loading the ovens with coal and for removing the coke, whereby these operations are effected more speedily and economically than in the ordinary manner. Ninth, in producing in the same ovens as those used for the manufacture of coke, common tar, such as that obtained from gas-works, or other tars richer in oils; and in utilizing them in various ways.

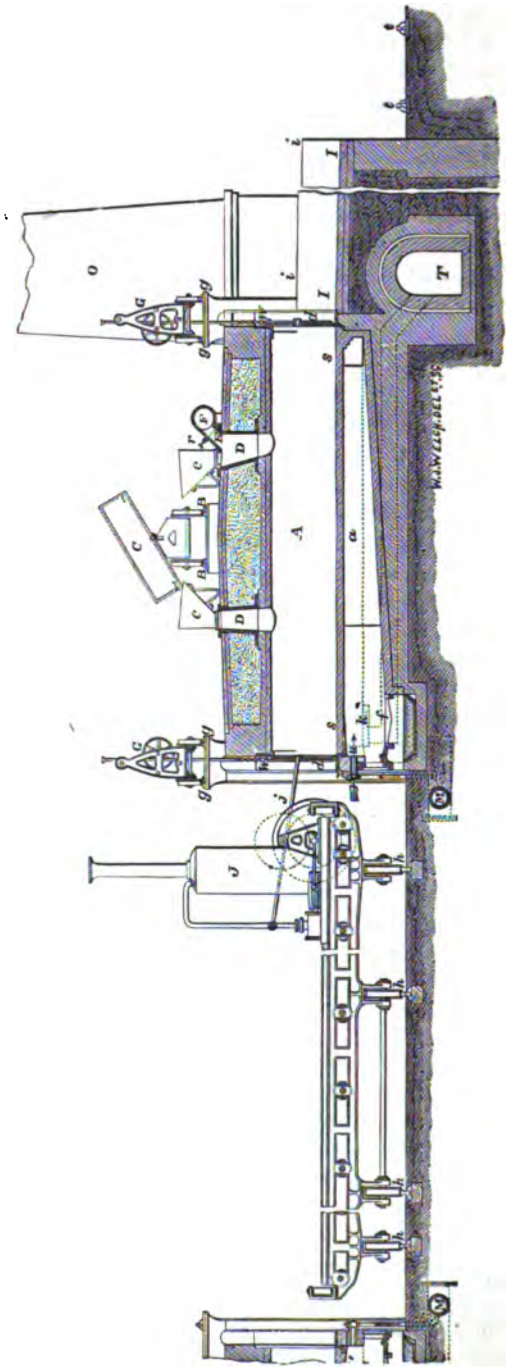

Fig. 18.—Pernolet's Coke-Oven; longitudinal section.

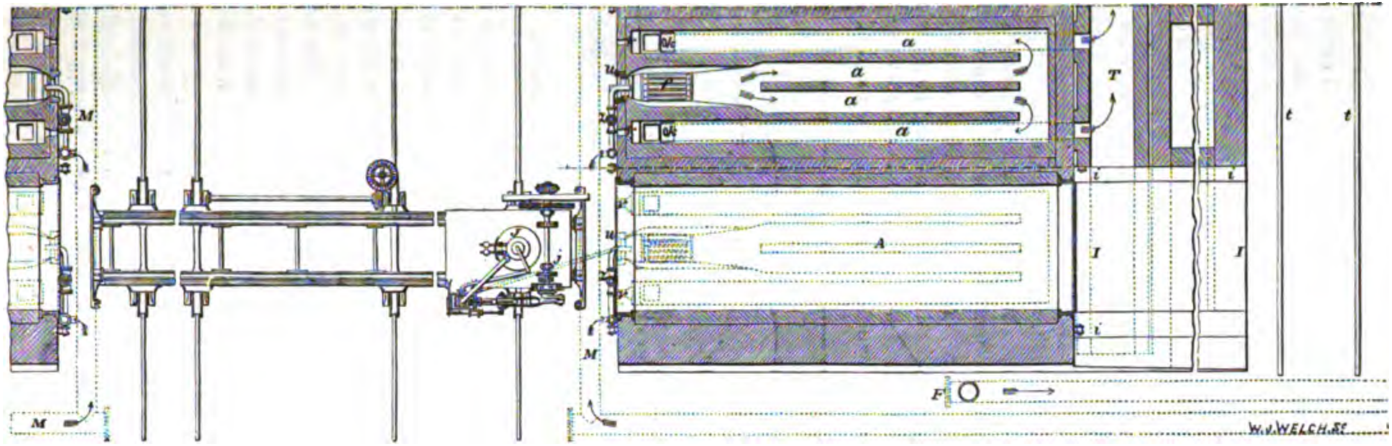

Fig. 19.—Pernolet's Coke-Oven; horizontal section at different heights.

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Ovens constructed according to this invention are equally suited for raw or for washed coal. The distillation is carried on upon a large scale, say from six to seven tons in an oven, and all the products accessory to the distillation, such as tars, ammoniacal waters, and gases for burning and heating, as well as the coke, being collected separately, may be utilized. If these ovens are erected near towns, the gas can be used for lighting, or it may be employed in iron-works, where a quick and regular fire is required. If the gas be not required for lighting or heating, it is burned in the coking-oven furnaces, in which case no other fuel will ordinarily be required. When the ovens are used for the manufacture of coke only, a gas-holder, which in other cases is required to contain the gas, is dispensed with, as well as the exhauster; and the exhaust will depend on the draught of the chimney, being increased by the vacuum produced by the partial condensation of the volatilized products.

The coke-oven shown, figs. 18, 19, and 20, is about thirty-two feet in length. Fig. 18 is a longitudinal section, and fig. 19 a horizontal section of two ovens at different heights. Fig. 20 is a transverse vertical section through two ovens; of one through the fire-place and of the other across the flues at the far end. C, fig. 18, is a truck running on the rails, B, and constructed in such a manner that its load of coal may be discharged at will into either of the openings, D, in the arch of the oven. An ordinary oven requires six truck-loads to fill it; *c*, are hoppers placed over the apertures, D, to guide the load into the interior of the oven, A. The coal being thus introduced on to the bed, *s*, of the oven, is spread evenly thereon by tools introduced at the ends. The two doors, *d*, are then lowered by means of the windlasses, G, running on the rails, *g*; the hooks, W, which connect the doors, *d*, with the windlasses, are disconnected, all the openings closed, and the joints stopped with clay, so that no air may enter the oven.

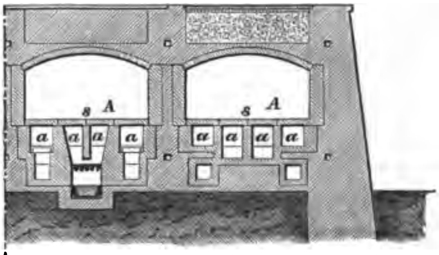

Fig. 20.—Pernolet's Coke-Oven; transverse section.

During the time occupied by the removal of the coke and charging the oven with coal, the fire on *f* has not been extinguished, and as soon as the mass of coal introduced into the oven is sufficiently heated, distillation begins. Gases, steam, and tar are disengaged and issue by the orifice E; they then traverse the pipe F, and enter the condenser, where the gaseous products traverse compartments formed of thin metallic plates close to one another, by coming in contact with which they are gradually cooled. The cooling is accelerated by a continuous flow of water falling from a reservoir into chambers, whence the water spreads on each side so as to run down externally from top to bottom over the sides of the condenser. A wooden casing surrounds the condenser, and protects it from the action of the sun, and at the same time affords a passage for a current of air, which serves to cool the water. Following the condenser, where the tar and greater portion of the ammoniacal waters are deposited, the gases and the ammoniacal vapours enter into the first washer, which they traverse, from the bottom upwards, through fine showers of ammoniacal water supplied continually by a pump from a suitable reservoir. The ammoniacal waters may be thus enriched to any desired degree for subsequent treatment. The current of gas thus purified passes to a second washer, similar to the first, but supplied with water only, to complete the absorption of ammonia. After the tar, oils, and ammoniacal waters have been removed from the gas, the latter is carried onwards through pipes, M, to the burners by which it is consumed. The combustible gases which reach the fire-place of the coke-oven, *f*, become mixed with air, which enters through small holes formed around and in the middle of the nozzle, *u*. The mixture of gas with air produces a flame which, circulating through the flues, *a*, conveys heat to the ovens. Thence the products of combustion pass to the flue, T, and afterwards to the chimney, O. The ovens are divided into sets of ten, each group having a separate chimney; *v*, are sight-holes, fitted to the flues, *a*, for examining whether the heat is equally spread between the different ovens of a group and in the various flues of the same oven. By means of a tap, *z*, the quantity of gas admitted to the fire-place of each oven is regulated. The quantity of air admitted around and through the nozzle, *u*, is regulated by valves or otherwise, and the distribution of flame is regulated in the flues, *a*, by means of fire-brick dampers, *k*. After a certain time, which varies from two to three days, according to the nature of the coal used, the whole of the charge is transformed into coke, and is ready for discharging.

For this purpose, a steam discharger is conveyed to the front of the oven on the rails, *h*; communication between the inside of the oven, A, and the general passage for the gas and vapours is intercepted by closing the damper, *r*; connection between the boiler, J, and the door of the oven is established by the pipe, *j*, and, by opening a tap, steam is driven into the interior of the oven as long as may be considered desirable. The doors, *d*, are then raised, and after adjusting the discharger in front of the oven, it is set in motion, and pushes the coke before it out of the oven on to the inclined floor, I, between dwarf walls, *i*, where it is covered with damp coke-dust to smother it. The discharging occupies from four to five minutes. The discharger is then withdrawn, the orifices, D, are opened, and the oven is again charged, as before, with from six to seven tons of coal, which takes from fifteen to twenty minutes to accomplish; the doors, *d*, are lowered, any pitch which may have been deposited is removed, the charging orifices are closed, the damper, *r*, opened, and the operation commences afresh.

When the coke on the inclined floor, I, is sufficiently cold, it is placed in trucks which convey it away on the rails, *t*. Care must be taken that the following precautions are attended to during the process. First, that all the joints are kept tight with clay, so that no air may enter during the operation. Second, that an uniform heat is maintained throughout the ovens. Third, that the condensers and washers are regularly supplied with water. The form and shape of the ovens may be considerably varied without departing from the invention. M. Pernolet states that impure coals may be pulverized in any convenient way, or mixed with rich coal, or even with tar, or pure coal before carbonization; the pulverizing process being very efficacious in order to give solidity and good appearance to the coke, and in order that coke made from very poor coal may, without washing, be suitable for use in locomotives and furnaces generally without leaving clinkers on the grate. Because, when the foreign matters are pulverized, the current of air carries them away, so that in this particular case the pulverization replaces washing, without waste. Where the mixture of coal to be formed into coke is composed of materials which do not carbonize well alone, and would produce only incoherent products, ovens constructed according to this invention answer better than ordinary ones, because the bituminous parts not being liable to be burnt, a less quantity is required to produce the agglomeration of poor coal, anthracite, coke-dust, and other combustible materials incapable, alone, of being transformed into solid coke.

The apparatus employed by Pernolet is very similar to that patented by Pauwels and Dubochet twelve years previously, excepting that the latter exhaust the gases from the ovens by means of a machine, whilst Pernolet sometimes effects the same object by drawing off the products of combustion through a high chimney.

*The Carvès Coke-Oven.*—In the earlier attempts to utilize the liquid bye-products resulting from the coking of coal, various difficulties were experienced, and for many years they were attended with very limited success. The waste products were from the first easily collected, but the coke was of inferior quality, and there was some trouble in maintaining the necessary flues. To M. Carvès belongs the credit of being the first to construct and work a coke-oven capable of yielding coke suitable for metallurgical purposes, coal-gas, ammoniacal liquors, and valuable coal-tar. The oven employed for this purpose is a horizontal chamber of brickwork about 32 feet in length, 2 feet in width, and 4 feet 6 inches in height. A number of these are built side by side, with a partition wall between them sufficiently thick to enclose three horizontal flues communicating with one another. Two flues are also formed under the floor of each oven, and in those originally constructed a small fire-place, consisting of a grate and ash-pit with a suitable door, was placed at one end. Over each fire-door is a nozzle, through which the gas generated during the process of coking, after having been deprived of its liquid products by condensation, is admitted and burnt over the ignited fuel lying upon the grate. The grates are charged twice only during the twenty-four hours, their purpose being simply that of keeping the gas ignited; but in the newer form of the Carvès oven the fire-places are omitted, and the only fuel employed is gas. In the *Simon-Carvès Oven*, in accordance with a recent improvement, hot air is introduced into the flues together with the gas, and as this air is heated by the waste heat from the flues which conduct the products of combustion to the chimney, a considerable increase of temperature is imparted to the ovens without any additional cost for fuel. From the burner, where the gas is consumed by the aid of heated air, the products of combustion pass forward through one of the horizontal flues beneath the bottom of the oven to its far end, and thence turn back through another, from which they ascend by a vertical passage into the uppermost of the horizontal flues in the partition wall, and descending in a zigzag course through those below it, finally pass into a flue outside the ovens, and thence to the chimney. When in operation, these ovens are hermetically sealed, with the exception of the opening for the escape of the volatile products, which are drawn off by an exhauster.

The coal, as in Pernolet's oven, is introduced through openings in the roof, over which iron trucks travel upon rails. These feed-holes are provided with iron covers kept tightly luted with clay during the process of coking, as are also the large doors at each end of the oven or retort. Through the middle of the arch of each oven rises a gas-pipe provided with a hydraulic valve through which the products of distillation escape into a range of cast-iron pipes kept cool by an external dripping of water over their surfaces, so that the principal part of the tar and ammoniacal liquors becomes condensed. The combustible gas thus separated from the condensed liquids is further passed through scrubbers containing coke moistened with ammoniacal liquor, which, after being repeatedly used, gradually becomes stronger until it reaches saturation, when it is run off and treated in the usual way. All the valuable bye-products having been thus withdrawn from the gas, it is led through iron pipes to nozzles, where it is consumed under the bottom of the ovens. The coke is removed by a steam-ram, similar to that employed for discharging the Pernolet oven.

Extensive ranges of ovens upon this principle have been erected by the Terre-Noire Company at Bessèges, France, and at various other Continental localities; the coke produced by them being, it is said, equal to that yielded by the older forms of coke-oven.

The production of coke is also larger than in the ordinary oven, since a coal producing from 58 to 60 per cent. of coke in the ordinary beehive oven will yield as much as 75 per cent. of a superior coke in the improved ovens. A range of ovens of this construction are in successful operation at the coking establishment of Messrs. Pease at Crook, near Darlington, where, in addition to 77 per cent. of coke of fair quality, and 2·60 per cent. of tar, 1·09 per cent. of ammonium sulphate is obtained from the coal treated. The time required to work off an oven is forty-eight hours.

*The Otto Coke-Oven.*—This, like the Carvès oven, is a modification of the Coppée coke-oven, in which heat is obtained by the combustion of the distillates in lateral flues, but which is not provided with apparatus for drawing off and collecting tar and ammoniacal liquors. The side-flues of the Otto oven, like those of the original Coppée, are vertical instead of being horizontal as in the Carvès oven, and the various liquid products are condensed and collected by an elaborate system of coolers. The time necessary for coking a charge is forty-eight hours, and the coke, when ready, is forced out of the oven by a steam discharger.

At the Chester meeting of the Iron and Steel Institute (1884), Dr. Otto stated that a successful attempt to construct coke-ovens in such a way as not only to produce coke but also to utilize the bye-products of coking, was made in France in 1862, while England and Germany have only begun within the last few years to construct ovens upon this principle. In the year 1884 about 150 ovens utilizing the bye-products of coking were working in France, 50 in England, and 190 in Germany; the latter to be shortly increased to 390.

*The Hoffmann Coke-Oven.*—The inventor of this system is Herr Gust. Hoffmann of Gottesberg, Silesia. Its essential features consist in the combination of coke-ovens with the Siemens regenerator in such a way as to heat the air serving for the combustion of the gas as highly as possible. In the ordinary coke-oven the heat necessary to maintain the coking is produced by the combustion of the gases either in the interior of the oven or in its side-flues. If the gases produced in a coke-oven are passed through a condenser, they lose their high temperature and return to the oven comparatively cold. It is only with coals rich in gas that it is possible to obtain a temperature sufficiently high for coking by burning the gases with cold air. In the best-constructed coke-ovens it is therefore sought to heat the air by which the gases are burned to the highest possible temperature, in order to restore the loss of heating power suffered by the latter while passing through the condensers. It is consequently the object of the Hoffmann system to bring the air at as high a temperature as possible in contact with the gases, since the hotter the air employed the greater will be the temperature obtained in the oven. Fig. 21 is a longitudinal section through the centre of a Hoffmann coke-oven, and fig. 22 a transverse section, at various points, through a portion of a block of these ovens.

In the Hoffmann oven there is no communication between the coking chamber, A, and the side-flues, H; these are not shown in fig. 21, although the openings leading to them are seen below the level of the hearth. In

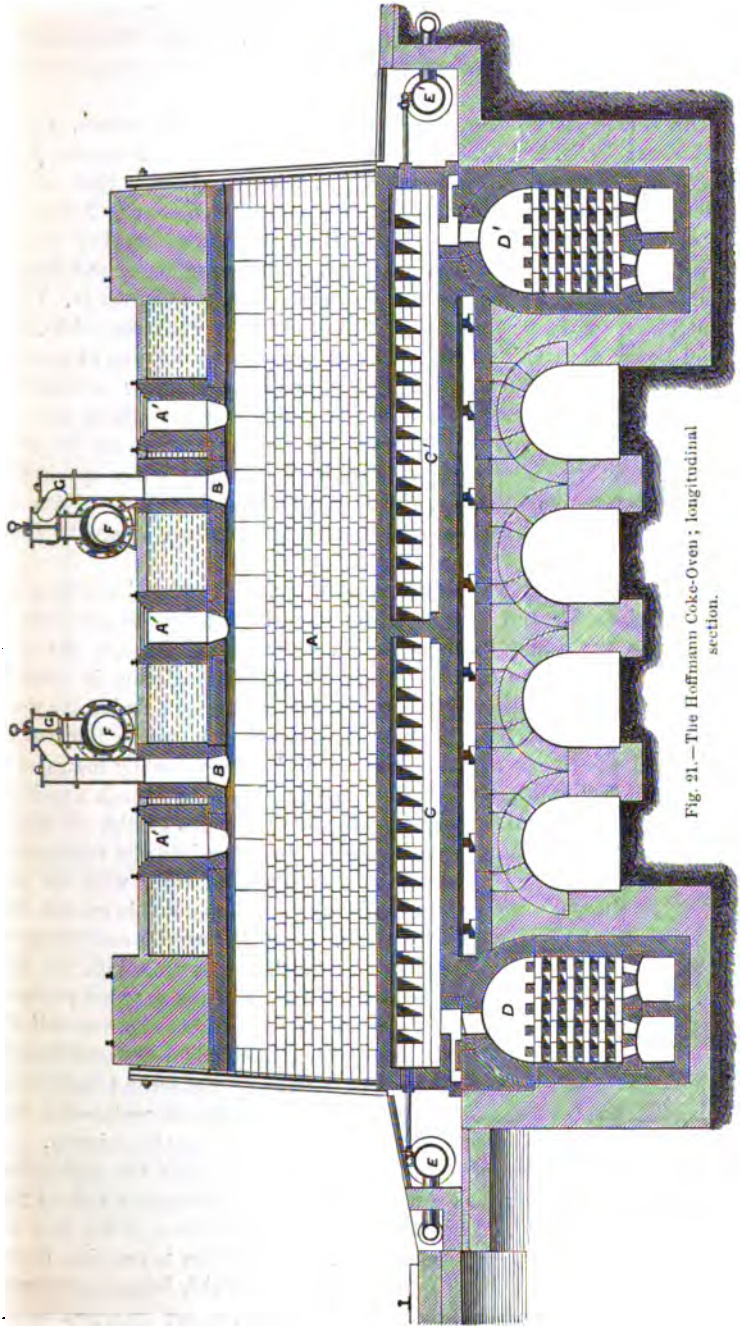

Fig. 21.—The Hoffmann Coke-Oven; longitudinal section.

fact, with the exception of the openings A′ for charging, and the doors for discharging, which are hermetically closed during the process of coking, there are only two openings, B, in the roof of the oven by which the gases escape.

The side walls of the ovens contain, over the abutments, a horizontal canal, I (fig. 22), which passes over the entire set of vertical flues, H, and is the means of communication between them. Each of the bottom flues is divided across into two equal lengths, C and C′ (fig. 21), each of these lengths communicating with a regenerator serving to heat the air for the combustion of the gases. The portion of the flue, C, communicates with the regenerator, D, and that marked C′ with D′. These regenerators are long flues on the Siemens plan with fire-bricks, and extend below the whole range of coke-ovens, communicating at one end by means of a clack-valve either with a pipe conveying air or with the chimney. The two gas-pipes, E, E′, extend along the whole range of coke-ovens from one end to the other. If now the ovens are supposed to be in operation, the gases from the charge escape by the openings B in the roof into the receivers, F, the valves, G, being open. From the receivers, F, the gases pass to the condensation house, where they are cooled and washed.

The gases returning from the condensers, where they have been deprived of their tar and ammonia, are passed back by the gas-pipes, E and E′, by the same machine which drew them through the condensers, and by means of which their complete circulation is effected. According to the position of a clack-valve in the gas main, the gases return either by the pipe E or by that marked E′, while the bottom flues of all the ovens communicate with the gas-pipes by means of a nozzle furnished with a tap. If the gas-valve is placed in such a position that the gas passes through the pipe E, then the valve of the air supply must be so disposed that the air is forced by a fan into the regenerator, D. The regenerator, D, and the nozzles communicating with the gas-pipe, E, discharge, respectively, heated air and inflammable gas into the bottom flues, C, of all the coke-ovens. In this way the combustion of the gases takes place in the flues, C, and thence extends into the side flues. The whole current of the burning gases and the very hot products of combustion thus rise by the vertical side flues from the first half, C, of the bottom flues into the horizontal flues, I, and then descend through the remainder of the vertical side flues, H, into the second half, C′, of the bottom flues, whence, after passing through the air regenerator, and imparting to it a portion of their heat, they escape by the chimney.

After the expiration of about an hour the position of the clack-valves in the gas-pipes and air-flues are changed, so that the course both of the gas and air is reversed. The gases are then forced through the pipe, E′, and the air through the regenerator, D′. Combustion takes place in the half, C′, of the bottom flues, and the side flues, which before served as a passage to the descending products of combustion, are traversed by an ascending draught; the products of combustion passing off through the air regenerator, D, into the chimney. The coke is removed from the oven by steam-power as in the case of the other coke-ovens belonging to his class.

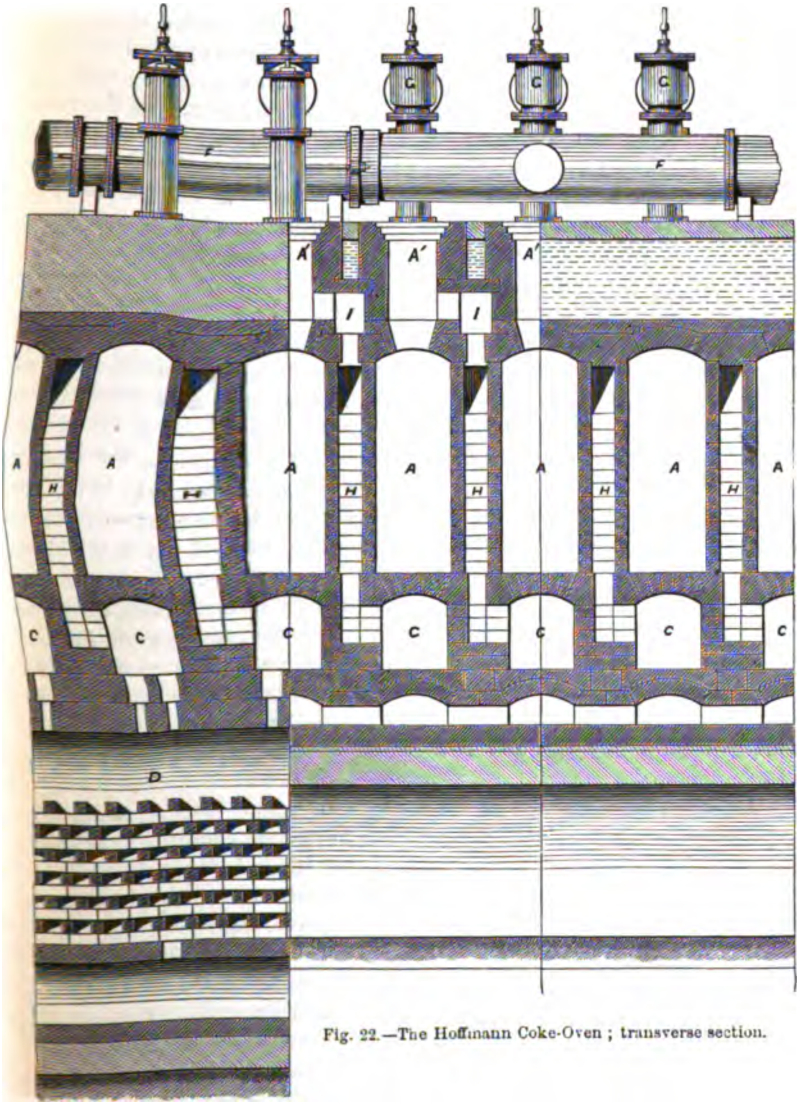

Fig. 22.—The Hoffmann Coke-Oven ; transverse section.

In one of the German establishments employing such coke-ovens, the daily production of gas per oven amounts to 24,700 cubic feet, whereof 17,700 only are required for coking purposes, thus leaving 7000 cubic feet daily available for other purposes. In these furnaces the bottom and side flues are so extremely hot that with charges of 8 tons 13 cwts. the coking process lasts only forty-eight hours.

The quality of the coke produced is stated to be equal to that obtained from ordinary coke-ovens, while its yield is several per cent. in excess of that made in the beehive oven. The yield and composition of the bye-products are much influenced by the character of the coal operated on. The coal of Westphalia yields 1 per cent. of ammonium sulphate, while in Upper Silesia this figure rises from 1·1 up to 1·7 per cent. In the Saarbrücken district, on the other hand, the coal yields only from 0·7 to 0·8 per cent. of this salt. The average yield of tar is about 3 per cent., but it does not contain so large a proportion of benzene as that produced in gas-works; the amount of naphthalene and anthracene is, however, about equal to that present in gas-tar. The cost of coking is said not to be greater than in the common beehive oven.

*The Jameson Coke-Oven.*—Mr. John Jameson of Newcastle-on-Tyne, instead of drawing off the gases and condensable products from the tops of his ovens, does so from the bottom, and collects and utilizes the bye-products from the ordinary beehive oven. The cost of converting an ordinary beehive oven in accordance with Mr. Jameson's system varies from £8 to £10, and to do so it is only necessary to replace the old bottom by a new one having radial depressions or gutters covered by perforated tiles, and sloping towards the centre, where there is an aperture in connection with a system of pipes and coolers exhausted by a Root's blower. Nine beehive ovens, at Page-Bank, have been converted in accordance with this system by Messrs. Bell, and produce coke which cannot be distinguished from that made in the same ovens previous to their conversion, while the yield is precisely the same. The time required for coking a charge is, with each system, seventy-two hours; and in addition to coke of the usual amount and quality, the Jameson ovens yield four gallons of oily tar, and two lbs. of ammonium sulphate per ton of coal operated upon.

Although practical men generally are fairly agreed that for use in the cupola, coke from which the bye-products have been collected is as good as that produced in the ordinary beehive oven, there is not such a general agreement with regard to its value for blast-furnace purposes. In a paper read before the Iron and Steel Institute, May 6th, 1885, Sir I. Lowthian Bell described a comparative experiment which he had conducted on a large scale with Bearpark coke made in ordinary beehive ovens, and the same coal coked in the Simon-Carvès ovens. This experiment would appear to show that the coke prepared in the ordinary beehive oven was, for blast-furnace purposes, about 10 per cent. more valuable than that made in ovens from which the bye-products had been collected.

COMPOSITION AND PROPERTIES OF COKE.—Well-prepared coke essentially consists of carbon, enclosing the various inorganic impurities of the coal from which it has been manufactured. It, however, invariably retains small quantities of hydrogen, oxygen, and nitrogen. The following table gives the composition of three varieties of coke:—

| | 1. | 2. | 3. |
|---|---|---|---|
| C. | 93·15 | 91·30 | 91·59 |
| H. | 0·72 | 0·33 | 0·47 |
| N. | 1·28 | 2·17 | 2·05 |
| O. | 0·90 | | |
| Ash | 3·95 | 6·20 | 5·89 |

No. 1, coke from Durham coal, analysed by Richardson; Nos. 2 and 3, coke from the caking coal of the Mons basin, analysed by M. de Marsilly. These specimens were dried at between 100° and 200° C. before being subjected to analysis.

Perfectly dry coke does not generally absorb more than 2·5 per cent. of water by exposure to a moist atmosphere, and coke, of which the extinction has been properly conducted, should not retain more than 3 per cent. of moisture. As, however, coke-burners are not unfrequently paid according to the weight of coke produced, without any stipulation being made as to the percentage of water it shall contain, it is not unusual to find specimens in which the amount varies from 8 to 12 per cent. It is evident that, in order to produce its highest calorific effect, coke should be used in a dry state. A considerable portion of the slack now employed for the manufacture of coke is freed from shale and other impurities by some process of washing before being charged into the ovens.

CHARRING BROWN COAL.—Brown coal is of all kinds of fuel the least adapted for carbonization; for although it is acted on by heat in the same way as wood, and produces a less combustible charcoal or coke, yet it is subject to inconveniences which render its production too costly for general application. Lignite, like peat, contains a large proportion of ash, and this percentage will necessarily be much greater in the coke produced than in the lignite from which it was made. This, from the tendency which the coke would necessarily have to clinker on the fire, prevents its being used for many purposes for which a fuel of greater purity could be employed. In addition to this, the action of heat causes the layers and concentric rings, which are scarcely perceptible in fresh lignite, to separate from one another, and the charcoal or coke manufactured is thereby either reduced to such small fragments as to be of little service as a fuel, or is rendered so extremely friable as to be unable to bear carriage even to a short distance from the locality in which it is produced. The carbonization of lignite in mounds has, however, been conducted on a small scale in the neighbourhood of Cassel; but the situations where this can be done with advantage are not numerous.

The following results were obtained by heating pieces of brown coal in closed crucibles until all traces of their volatile constituents had ceased to be evolved :—

| 100 Parts of | Gave of Charcoal |
|---|---|
| Earthy Coal from the Basses Alpes | 48·5 |
| Lignite from Greece . . . | 38·9 |
| „ Friesdorf . . . | 28·2 |
| „ Iceland . . . | 57·5 |
| „ Auszig . . . | 40·1 |
| „ Orsberg . . . | 62·8 |
| „ Hegendorf . . | 41·2 |
| „ Stöszchen . . | 29·1 |
| „ Pützchen . . . | 46·4 |
| „ „ . . . | 44·7 |
| Lignite from Neundorf . . . | 38·4 |
| „ Coulang . . . | 38·1 |
| „ Jahnsdorf . . | 32·3 |
| „ Hartenberg . . | 37·2 |
| „ „ . . | 34·6 |
| „ Kanden . . . | 37·5 |
| „ Reichenau . . | 38·1 |
| „ „ . . | 29·3 |
| „ Altsattel . . . | 40·3 |
| „ Verau . . . | 35·6 |

## GASEOUS FUEL.

Attempts were made as early as the commencement of the present century to substitute combustible gases for solid fuel in various technical operations; but, for a considerable time, they were not attended with practical success. According to a report, made in 1842 to the French Academy, by Thénard, Berthier, and Chevreul, the waste gases of blast-furnaces were first employed by M. Aubertot in the year 1809 for roasting ores, burning lime, &c., and in 1814 he suggested the erection of suitable apparatus for the employment of waste gases for metallurgical purposes. In the year 1801 Lampadius had already shown the possibility of employing the waste gases from the carbonization of wood, and in 1830 he attempted, at smelting-works near Freiberg, the cupellation of argentiferous lead by means of gases produced from coal.

Ebelmen states that M. Victor Sire, of Clerval, obtained a patent in 1836 for the manufacture of wrought-iron by means of waste gases from blast-furnaces. According to a report of the Central Jury of the Paris Exhibition in 1844, Sire's patent was employed in 1838 at iron-works on the Lower Rhine, and in 1841 in the Department of the Moselle, for the refining of iron; but the process does not appear to have attracted much attention.

Successful experiments were made in 1837 by Wilhelm v. Faber du Faur, by burning the waste gases of blast-furnaces in a reverberatory furnace for the purpose of puddling pig-iron.

In the course, however, of the various trials of the employment of waste gases for this purpose, it was soon discovered that every modification in the working of the blast-furnace produced a corresponding change with regard to the quantity and composition of the gases evolved, and that the process of puddling was thereby prejudicially affected. The collection of the gases also appeared to produce a prejudicial effect on the operations of the blast-furnace itself. The dependence for a supply of fuel on the satisfactory working of the blast-furnace was found so prejudicial as to cause this method of employing waste gases to be abandoned, and led to their employment for the roasting of ores, the heating of the blast, the production of steam, and the burning of lime, bricks, &c. These processes do not require either a very high or a very uniform temperature, and a large amount of fuel was thus saved. The utilization of waste gases has also resulted in the extensive employment for metallurgical purposes of gases specially prepared in generators or producers, by which means fuels of an inferior description can be rendered serviceable for purposes for which, if consumed in the ordinary way, they would be totally unfitted.

Before Faber du Faur's process for utilizing waste gases had obtained publicity, experiments were made in the Harz (1839) by Bischof, with the view of generating gases in a furnace or producer, and subsequently burning them by means of atmospheric air.

Gases thus obtained from peat were found to readily afford the highest welding heats, but as Faber's method of employing waste gases had in the meantime come into notice, and appeared to possess the advantage of requiring no special consumption of fuel, Bischof's results were not at once appreciated. In 1838 some Austrian metallurgists who visited Wasseralfingen, in Würtemberg, where Faber's process was in operation, came to the conclusion that the employment of waste gases in the puddling of iron could not afford practical results, and in the following year commenced experiments at the iron-works of Jenbach in the Tyrol, with a view to the preparation of inflammable gases by an imperfect combustion of small charcoal. These trials, however, gave rise to dangerous explosions which appear to have finally led to their abandonment. In 1841 Karsten stated it was probable that certain descriptions of compact fuel, which, from their state of aggregation and low calorific power, were not then adapted for the purpose of puddling, would, ere long, be rendered available for that operation by being converted, in a special apparatus, into carbon monoxide gas, by the combustion of which the desired result would be effected. The experiments made at Jenbach having shown the practicability of firing with artificially produced gases, further experiments were made in 1842 at steel-works at St. Stephan, Styria, with a view of producing gaseous fuel from small brown coal. The results obtained having been of a satisfactory nature, they were at once published, and gave rise to the general introduction of artificially prepared gases as fuel. This method of firing has been further developed by Bischof, Eck, and others, but more especially by Sir C. W. Siemens, whose regenerative gas-furnace is suited for almost every metallurgical operation in which it is required to produce a high temperature in reverberatory furnaces. As this may be regarded as one of the most important inventions of modern times with regard to the utilization of fuel, it will be necessary to describe not only the apparatus employed for the preparation of the gases, but also the arrangements for effecting their economical combustion.

Among the many advantages claimed by Siemens for his regenerative furnaces were the following:—

*a.* The employment of inferior descriptions of fuel, such as slack, coal-dust, lignite, peat, &c., together with a saving on the quantity made use of to the amount of from 40 to 50 per cent.

*b.* A daily increase of the work done in a furnace of given dimensions, amounting to at least 30 per cent.; which is a result of the almost unlimited calorific power at command, even when only a moderate chimney-draught is available.

*c.* Perfect uniformity of heat throughout the furnace, and greater durability of the brickwork; owing to the absence of ashes, by which the fusibility of the surfaces with which they come in contact is increased.

*d.* The production of a flame of such purity as greatly to diminish waste by oxidation or deterioration of the metals operated on.

*e.* Great cleanliness and saving of space in works; since the gas-producers are invariably erected on the outside.

*f.* Increased command of the heat employed and of the chemical effects produced by the flame; which can be immediately checked when required, or at once changed from an oxidizing to a reducing one, or *vice versâ.*

*g.* Absence of smoke from the chimney-stack; which, in the neighbourhood of large towns, and in some other situations, is of great importance.

THE SIEMENS GAS-PRODUCER.—An early form of the gas-producer employed by Messrs. Siemens is shown, figs. 23 and 24. Fig. 23 is a vertical section, and fig. 24 a plan, partly section on the line P, Q. The body of the apparatus, A, is a rectangular fire-brick chamber, of which the side, B, consists of thick cast-iron plates, lined with fire-brick, and having a step-grate formed of flat iron bars, *b.* At bottom the bars, C, forming the grate, are of wrought-iron, two inches square, and rest on suitable cast-iron bearers built into the masonry. The fuel employed for the production of gas is, in this country, bituminous coal, which should not possess the property of caking in too high a degree, but coke, lignite, peat, and even sawdust, may, in case of necessity, be used for the same purpose. The fuel, whatever may be its nature, is charged into the hoppers, D, and, on opening the valves in connection with the weighted levers, *d*, falls on the inclined plane forming the front of the producer. Before doing this, however, the top of the hopper, from which a charge is about to be let fall, is closed by an iron lid to prevent the escape of gas during the operation. In this way the grate is constantly kept thickly covered with fuel, and the accumulated ash and clinker are occasionally withdrawn by removing the bars, C, beneath which they are allowed to accumulate for the purpose of conveniently regulating the admission of atmospheric air. During the removal of the grate, C, for the purpose of clinkering, temporary pointed bars of wrought-iron are inserted over the lower bars, *b*, and allowed to rest on the brickwork at

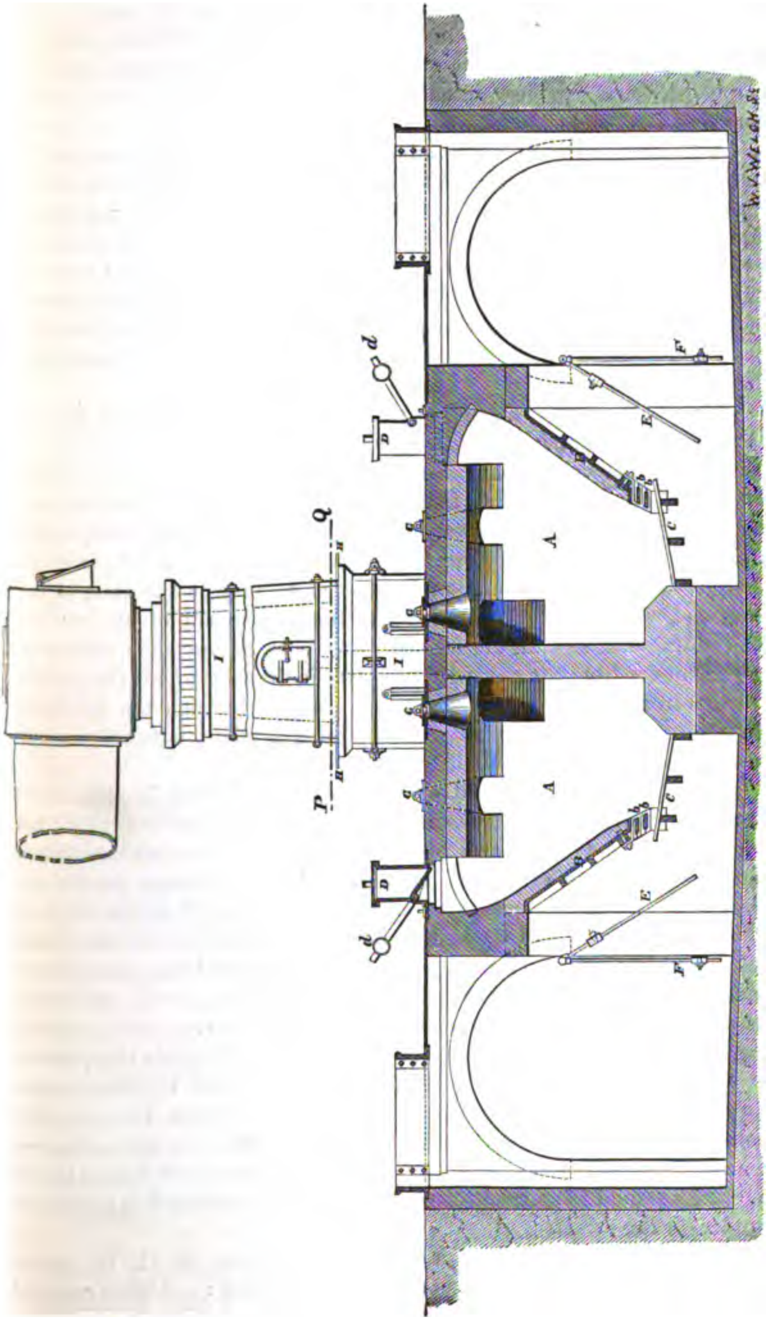

Fig. 23.—Siemens' Gas-Producer; vertical section.

the back. In this way the fuel in the cavity of the producer is supported, so that the ash and clinker may be removed without any interruption to the working of the apparatus. When coal of good quality is employed, each producer usually requires clinkering but once in the course of forty-eight hours.

A limited supply of air is admitted at the grate, and its oxygen, by uniting with the carbon of the fuel, forms $CO_2$, which rises through the ignited mass, taking up an additional atom of carbon, and thus giving rise to the formation of 2CO. The heat thus produced distils off hydrocarbons and other gases and vapours from the fuel as it gradually descends towards the grate, whilst the CO, diluted by the nitrogen of the air, and by a small quantity of unreduced $CO_2$, mixed with the gases and vapours distilled from the raw fuel, is finally conducted by a flue to the furnace.

A pipe, E, supplies a small quantity of water to the ash-pit, whence, as it evaporates and comes in contact with the incandescent fuel, it becomes decomposed, giving rise to carbonic oxide gas and hydrogen. The hose, F, of vulcanized indiarubber, is employed for watering the clinkers as they are withdrawn, in order that they may be rapidly cooled and their removal from before the producers facilitated. By means of the plug-holes, G, the workman is enabled to inspect the state of the fires, and, when necessary, to stir the fuel by the aid of an iron bar. The sliding dampers, H, are for the purpose of, at any time, cutting off the producers from the gas-stack, I. Any in-draught of air through the crevices in the brickwork, which would result in burning the gas in the flue, is prevented by constantly maintaining a slight outward pressure in the gas-channel.

When the furnaces stand on a considerably higher level than the producers, the required pressure is obtained without difficulty; but when this is not the case, a special arrangement becomes necessary in order to produce this effect. The most simple contrivance for the purpose is a cooling-tube raised to a considerable height above the level of the producers. The stack, I, is carried up in brickwork, well bound with iron, to a height of from ten to twelve feet, and the gases are conducted through a horizontal tube of wrought-iron, from which they pass down, through a similar stack of masonry, to the main gas-flue in connection with the furnaces. The gases, which rise from the producers at a temperature of about 550° C., are thus cooled by their passage through the metallic tube, and the descending column, becoming more dense and heavier than that of the same length which is ascending, consequently overbalances it. A syphon is thus formed, of which the two legs are of equal length, but of which one is filled with a heavier gaseous fluid than the other.

Shortly after the application of this arrangement, Sir C. W. Siemens made the following observations relative to its action:—"This method of obtaining a pressure in the gas-flue by cooling the gas has been objected to as throwing away heat that might be employed to greater advantage in the furnace; but this is not the case, because the action of a regenerator is such that the initial temperature of the gases to be heated has

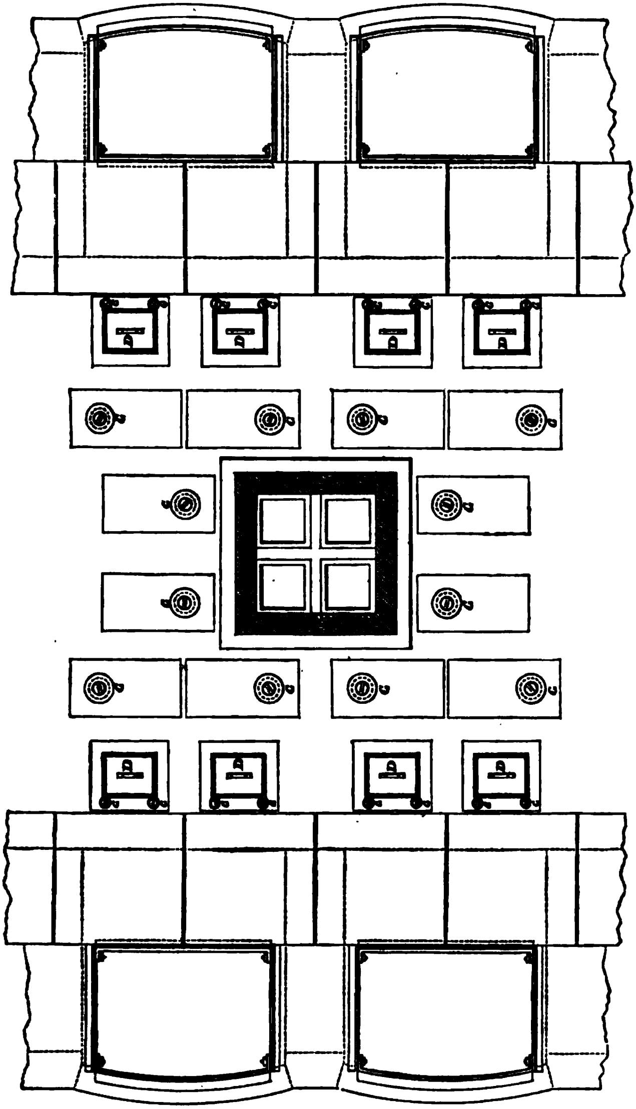

Fig. 24.—Siemens' Gas-Producer; plan, partly section, on P, Q.

no effect on the final temperature, and only renders the cooling of the hotter fluid more or less complete. The only result, therefore, of working the furnace with gas of higher temperature is to increase the heat of the waste gases passing off by the chimney-flue. The complete cooling of the gas results, on the other hand, in the great advantage of condensing the steam that it always carries with it from the gas-producer, and, in the case of iron and steel furnaces, in burning wet fuel, it is absolutely necessary to cool the gas very thoroughly in order to get rid of the large amount of steam that it contains, which, if allowed to pass into the furnace, would oxidize the metal.

"There is undoubtedly a certain waste of heat, which might be utilized by surrounding the cooling-tube with a boiler or by otherwise economising the heat it gives off, as, for instance, in drying the fuel; but the saving to be effected is not very great, for as 100 volumes of the gas require for combustion about 130 volumes of air, including 20 per cent. above that theoretically required, the heat given off in cooling 1000° F. is no more than would be lost in discharging the products of the complete combustion of the fuel at a temperature 435° in excess of the actual temperature of 200°; and this loss is greatly diminished if a richer gas is obtained."

He further observes that the composition of the gases varies in accordance with the nature of the fuel employed and the management of the gas-producer. An analysis made of the gas from the producers at the plate glass-works of St. Gobain, France, which were supplied with a mixture of three-fourths caking coal and one-fourth non-caking coal, afforded the following results:—

| | Volumes. |
|---|---|
| CO | 23·7 |
| H | 8·0 |
| $CH_4$ | 2·2 |
| $CO_2$ | 4·1 |
| N | 61·5 |
| O | 0·5 |
| | 100·0 |

The trace of oxygen present is no doubt due either to want of care in collecting the gas, or to leakage of air into the flue; allowing for this, the corrected analysis will stand as under:—

| | Volumes. | |
|---|---|---|
| CO | 24·2 | 34·6 |
| H | 8·2 | |
| $CH_4$ | 2·2 | |
| $CO_2$ | 4·2 | 65·4 |
| N | 61·2 | |

The first three of these constituents, or about 35 per cent., are alone of any use as fuel; the carbonic anhydride and nitrogen present only dilute the inflammable gases.

Regenerative Furnace.—In the regenerative furnace the producer-gases and air employed for their combustion are separately heated by the waste heat of the flame by means of regenerators placed beneath the furnace. These consist of four chambers fitted with fire-bricks loosely stacked together, so as to expose the largest possible surface to the gases passing between them. The waste gases from the furnace above are drawn down through two of these regenerators, heating the upper rows of bricks to a temperature little inferior to that of the furnace itself, and, passing successively over cooler surfaces, finally escape to the main flue of the chimney in a comparatively cold state. The direction of the draught is now reversed, and the flame and heated waste gases are employed to heat up the second pair of regenerators; at the same time the gases and air entering the furnace are passed in an opposite direction through the first pair, and coming, at first, in contact with the cooler brickwork below, are gradually heated in their ascent until, on arriving near the top, they attain a temperature nearly equal to that possessed by the products of combustion.

The heated gases and air finally pass up into the furnace, where they meet and ignite, producing a strong flame, which, after passing through it, is drawn down the second pair of regenerators to a flue in connection with the chimney. In this way the temperature of the ascending gas and air remains nearly constant, until the brickwork of the upper portion of the regenerator has sensibly cooled; but, by the time this has taken place, the second pair of regenerators have become sufficiently heated and the draught is again reversed. The current of waste gases is thus made to circulate through the first pair of regenerators, by which they again become heated, whilst the combustible gases and air, entering the furnace, are being passed up through the second pair. By reversing, in this way, the direction of the draught at regular intervals, a large proportion of the heat generated is retained in the furnace, whereas, under ordinary circumstances, a very considerable amount would be carried off by the products of combustion. Thus, whatever may have been the intensity of the heat produced in a regenerative furnace, the temperature of the gases escaping to the chimney rarely much exceeds 150° C.

In ordinary furnaces the amount of heat carried off by the products of combustion is often in excess of that utilized, since all the heat below the required temperature is entirely lost. The economy of fuel effected in the regenerative furnace, by removing this source of loss and making the heat of the waste gases available for raising the temperature of the flame, amounts, on an average, to 50 per cent. on the quantity used in an ordinary furnace, and this saving is greater in proportion as the temperature at which the furnace is worked is increased.

When the heat of a furnace is not continually lowered by the introduction of fresh charges of cold materials, the temperature for some time increases after each reversal of the direction of the draught, as a comparatively small proportion of the heat generated is carried off by the waste gases. By ascending through the regenerators the gases and air become heated to a temperature nearly equal to that of the flame which had been previously passing through them in a contrary direction, and, when they meet and burn in the furnace, the heat of combustion is added to that absorbed in the regenerators. The flame produced is consequently hotter than previously to the last reversal, and raises the regenerator through which it is passed to a greater heat than before. On again reversing the draught, this increased heat is communicated to the entering air and gases, and a still further increase in the temperature of the flame is the result. The temperature that may, in this way, be ultimately obtained by the gradual accumulation of heat in the regenerators and furnace is, in practice, limited by the difficulty of finding a sufficiently refractory material for the construction of the apparatus.

One of the best materials available for this purpose are Dinas fire-bricks, consisting of nearly pure silica; but although they perfectly withstand the temperature required for the fusion of the most refractory steel, the heat can, nevertheless, be so increased as to melt them.

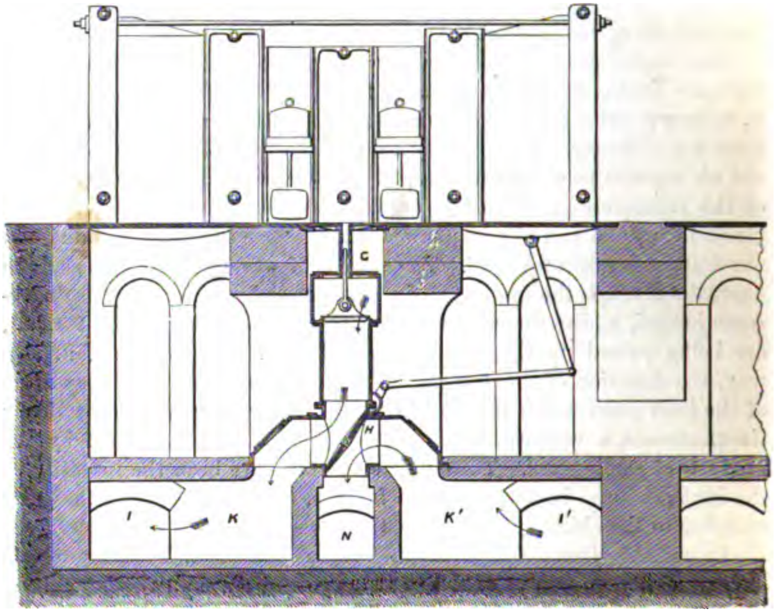

Fig. 25.—Siemens' Re-heating Furnace; front elevation, valves and flues in section.

It was calculated originally that, supposing the direction of the draught of a furnace to be reversed every hour, 17 lbs. of regenerator brickwork, at each end of the arrangement, per lb. of coal consumed in the gas-producer per hour, would be theoretically sufficient to absorb the waste heat, if the whole mass of the regenerators were uniformly heated at each reversal to the full temperature of the flame, and then completely cooled by the air and gases coming in. In practice, however, this does not actually take place, and, consequently, three or four times as much brickwork is required in the regenerators as is equal, in regard to capacity for heat, to the products of combustion. It has been found by experiment that a surface of six square feet in the regenerators is necessary to take up the heat of the products of combustion of 1 lb. of coal per hour. The arrangement of a reverberatory furnace on the regenerative principle will be understood by the aid of figs. 25, 26, and 27.

The first is a front elevation of a re-heating furnace, showing the gas-reversing valves and flues in section. Fig. 26 is a longitudinal section on A, B, C, D (fig. 27), and fig. 27 is a sectional plan on L, M (fig. 26). The waste heat escaping from the furnace is arrested and absorbed by the masses of open brickwork, E, F, E′ F′ (fig. 26), while the products of combustion subsequently reach the chimney-flue in a comparatively cool

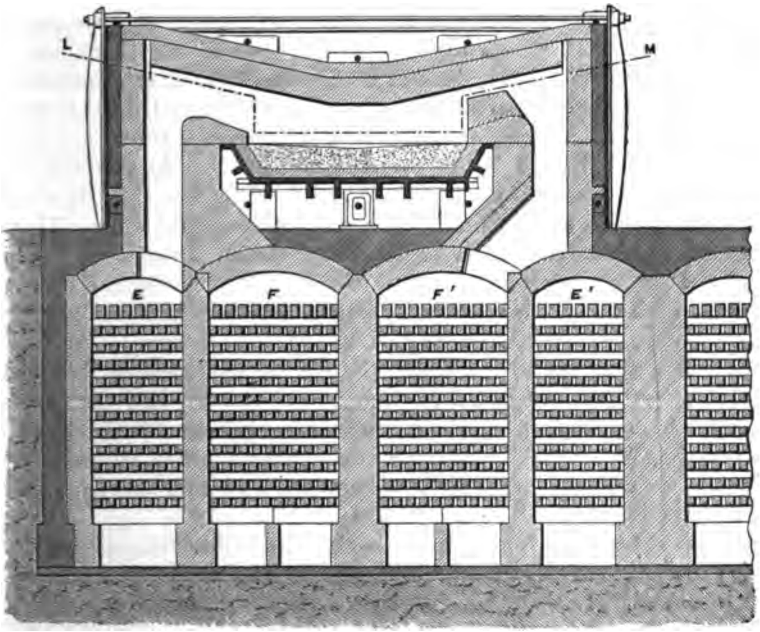

Fig. 26.—Siemens' Re-heating Furnace; longitudinal section.

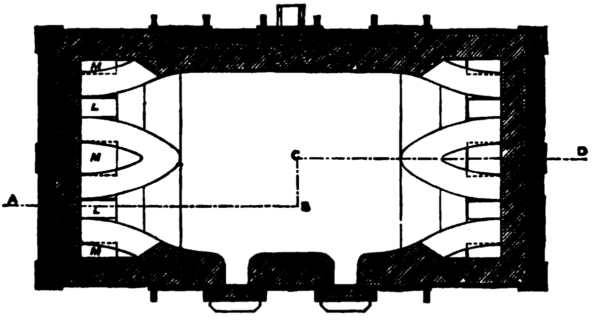

Fig. 27.—Siemens' Re-heating Furnace; sectional plan on L, M.

state. On first lighting the furnace, the gases pass in through the gas regulating valve, G, and the reversing valve, H (fig. 25), and entering the flue, I, reach the bottom of the regenerator, E (fig. 26). The air enters through corresponding air regulating and reversing valves behind the valves, G, H, and passes thence through a flue behind the partition, K, into the regenerator, F. The currents of unheated gas and air ascend separately through the regenerators E and F, and pass up respectively through the flues, L, M (fig. 27), into the furnace above, where they burn with a flame of moderate calorific intensity. The products of combustion are removed through a similar set of openings at the other end of the furnace, and passing into the regenerators, E′, F′, finally escape through the flue, I′, and another behind K′, to the chimney-flue, N. In this way the waste heat is absorbed by the brickwork of the regenerators, E′, F′, whilst the gases pass off to the chimney in a comparatively cold state. At the expiration of about an hour the reversing valves, through which the gas and air are separately admitted to the furnace, are turned over by means of levers, and an inward current is established through the regenerating chambers, E′, F′, which have become heated by the flame and waste gases which, previous to the reversal of the valves, had been descending through them. The air and gases now entering the furnace become heated in their passage through the hot brickwork of E′, F′, and, on meeting and entering into combustion, produce a higher temperature than that obtained during the first hour, when cold air and gases only were supplied.

The waste gases from this combustion at a higher temperature now heat the previously cold regenerators, E, F, to a correspondingly increased degree. After about an hour's working of the apparatus with the current in this direction, the reversing valves are again turned, and the air and gases admitted through the chambers, E, F, which are now very hot, and they consequently enter the furnace at a still higher temperature than before, producing a heat of increased intensity, and heating the regenerators, E′, F′, to a still higher temperature than E, F. On reversing the current, the air and gases acquire a greater heat than before, and an accession of heat is thus, step by step, obtained, until the furnace acquires the temperature required; the heat of the products of combustion being always so abstracted that they arrive at the chimney-flue in a comparatively cool state. When the required heat has been acquired by the furnace, its temperature is readily controlled by the supply of air and gas admitted through the regulating valves, and by the chimney damper, which is more or less closed as circumstances may require. The regulating valves are raised and lowered either by a hand-screw or by a notched lever, and consequently admit of being readily maintained in any required position.

Although furnaces constructed upon the regenerative principle vary considerably in their minor details, their main features remain in all cases essentially the same. In a recent paper contributed to the

'Journal of the Iron and Steel Institute'[1] by Mr. F. Siemens, he calls attention to the fact that to obtain the most satisfactory results from the regenerative furnace the dimensions of the reverberatory chamber should not be too much restricted.

In all furnaces destined for the production of a very high temperature, it has been generally considered, as the first condition of success, that the space to be heated should be as small as the particular circumstances of the case will allow, in order that the flame may be brought into immediate contact with the refractory lining, and more particularly with the surface of the material lying on the furnace bottom. After numerous and careful trials, Mr. Siemens has become satisfied that in order to obtain the best results the gas-furnace should be so arranged that the flame only radiates on the material to be treated, and should not come in direct contact with it, as was the case in the older furnaces. The gas- and air-ports, instead of being so situated that the flame impinges on the material placed upon the bottom of the furnace, should open a little below the roof of the melting chamber, and also at some distance from its side walls. In this way, after ignition, the gas and air have unobstructed space for entering into combustion and for the free development of the flame, which must be kept as much as possible from contact with either the lining of the furnace or with the material to be heated. It is also important that when crucibles, blooms, ingots, or piles of iron are operated upon, they should be placed sufficiently far apart to allow the radiant heat to have free access all round them. When the furnace is provided with working doors, the flame should pass above their level, as in that way the workmen will not be inconvenienced when watching the operations within the heating chamber. The regenerating furnace, as now constructed, appears, at first sight, to be unnecessarily wide and high, but by this method of heating high-temperature furnaces by radiation, considerable space is required for the development of the flame.

Numerous examples of the efficiency of this improved form of gas-furnace are quoted by the author of the paper, but one of these only will be here sufficient. At the Landore Siemens-Steel Works, near Swansea, the furnaces, which were altered at the end of 1883 and beginning of 1884, had before the end of September in the latter year already outlasted furnaces reconstructed at the same time on the old type. They work more uniformly, consume less fuel, and, on account of the smaller amount of oxidation going on, yield a larger out-put than the older form of furnace. This improved method of heating has been adopted in several steel-melting establishments in this country, and is particularly recommended when steel is to be prepared from an inferior quality of pig-iron by the adoption of a basic lining.

WATER-GAS.—Instead of using gas obtained from gas-producers of the ordinary description, a gaseous fuel known as "water-gas" has been recently sometimes employed. This gas is obtained by the action of superheated steam upon coal, a portion of the fuel being burnt as completely as possible into carbonic anhydride and water, while the resulting heat is stored up partly in the unconsumed fuel and partly in regenerators constructed on the Siemens principle. The heat accumulated in the regenerators is employed for the preparation of superheated steam, which is subsequently passed through the heated fuel, by uniting with the constituents of which carbonic oxide and hydrogen are produced and employed as fuel.

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[1] 'On a New Method of Heating the Regenerative Gas-Furnace,' by Frederick Siemens, C.E. Jour. Iron and Steel Inst., 1884, p. 434.

Gas-producers upon this principle have been erected at various localities in this country by Mr. W. S. Sutherland of Birmingham, who employs a system invented in America by Mr. Strong; a similar apparatus has likewise been constructed at Essen, in Germany. All these are reported to work more or less satisfactorily, but the results which have been hitherto published do not appear to be of a sufficiently detailed character to enable a comparison to be established between water-gas and the gas obtained from the Siemens producer. With a producer on the new principle, Mr. Sutherland states he has obtained 55,000 cubic feet of gas from one ton of Staffordshire coal, or about 25 cubic feet per pound of fuel consumed.

The composition of this gas was, by volume, approximately as follows:—

57 parts H and hydrocarbons.
35 ,, Co.
8 ,, $Co_2$ and N.
100

As compared with illuminating gas, Mr. Sutherland estimates the heating power of water-gas at about one-half, but, on the other hand, it can be prepared at very much less cost. By passing the gas from the producers through suitable condensers, about 1¼ gallon of tar and a certain quantity of ammoniacal liquor, of 4° Twaddell, have been obtained from a ton of coal.

## REFRACTORY MATERIALS FOR FURNACES AND CRUCIBLES.

**Fire-Stones.**—Many varieties of rock, rich in silica, have been employed for the refractory linings of furnaces, and before being used for that purpose should be stored for a considerable time in a dry place, in order to deprive them of moisture. When stratified rocks are used, they should be built into the wall in accordance with their natural bedding, in order, as far as possible, to prevent exfoliation on the application of heat. Sandstones, in which the grains of quartz are cemented by a siliceous or argillaceous cement, are sometimes employed for the hearth-stones of blast-furnaces, but those varieties which contain notable quantities either of lime or of iron oxide are not sufficiently refractory. Sandstones containing spangles of mica or grains of iron pyrites are not, generally, sufficiently infusible to be so employed; coarse-grained siliceous stones, such as Millstone-grit, are frequently made use of for this purpose.

In the Truckee Valley, State of Nevada, United States of America, a diatomaceous fire-stone is obtained, which presents the appearance of yellowish chalk. It is, however, much lighter, floating readily when first thrown into water, but sinking as soon as it becomes wetted to a certain depth. Sp. gr. = 1·49. It cuts readily, and may be sawn into any required form; it is also easily worked with the axe.

A specimen of this substance, analysed by A. G. Phillips, afforded the following results:—

| | 1. | 2. |
|---|---|---|
| $SiO_2$ . . . . . | 73·32 | 73·32 |
| $Fe_2O_3$ . . . . . | 3·23 | 3·18 |
| $Al_2O_3$ . . . . . | 9·48 | 9·69 |
| $CaO$ . . . . . | 0·70 | 0·82 |
| $K_2O$ . . . . . | 0·55 | 0·42 |
| $Na_2O$ . . . . . | 0·62 | 0·68 |
| $H_2O$, combined . . | 7·53 | 7·49 |
| $H_2O$, hygroscopic . | 4·65 | 4·65 |
| | 100·08 | 100·25 |

Talcose slate and soapstone have been sometimes used for the firework of furnaces, and, from the resistance offered by them to the corroding influence of metallic oxides, they are occasionally found very serviceable. Serpentine, on account of the small proportion of silica it contains, is also less readily attacked by metallic oxides than more siliceous rocks; it is used in Styria for the hearths of blast-furnaces. Gneiss is used at Schmöllnitz for the construction of reverberatory furnaces; it is easily dressed to the required forms, and offers great resistance to high temperatures and sudden changes of heat. Granite is employed in Cornwall for the outside masonry of lead and tin furnaces, which are internally lined with fire-brick. Siliceous sand is extensively used by the metallurgist, both for mixing with the fire-clay employed as a cement in the construction of furnaces and for furnace-bottoms. The beds of the smelting and refining furnaces in copper-works are made of a sand of which great accumulations are met with at various points along the coast of Great Britain. This sand usually contains about 87 per cent. of silica, with a little lime, oxide of iron, and alumina. Sand is also extensively employed for moulds both in iron and brass foundries.

FIRE-CLAYS.—Clays are essentially hydrated silicates of aluminium, and on the presence of water of combination depends their plasticity or capability of being moulded into any required form, when mixed with a suitable amount of water. In addition to their combined water, clays contain a greater or less amount of hygroscopic moisture, which may be expelled by heating them to a 100° C., without impairing their plasticity. When clays possess the property of resisting exposure to high temperatures, without either melting or becoming sensibly softened, they are called *refractory clays* or *fire-clays.* These occur in various geological cal formations, but more particularly in the Coal-measures.

When clays are heated to redness, their water of combination is expelled, as well as their hygroscopic water, and their property of affording, when mixed with water, a plastic mass is thereby destroyed. In this dehydrated state clays do not immediately combine with water, although they absorb it with avidity, without, however, regaining their plastic condition.

The substances found accompanying clays and in a state of intimate mixture with them are very numerous; but among the most common of these impurities may be mentioned oxide of iron, calcium carbonate, and iron pyrites. The presence of these substances materially impairs the refractory quality of the clays in which they occur, as, by their action on the silica, a series of fusible vitreous compounds is produced. The degree of heat necessary to effect the fusion of these compounds is not entirely dependent on the amount of extraneous matter thus brought in contact with the natural silicate, but is also in a remarkable manner influenced by their nature and number.

In this way a clay containing a given amount of magnesia is found to be less fusible than another similarly constituted, but having a certain portion of the magnesia replaced by an equivalent amount of lime; if iron oxide be also present, the compound will be found proportionately more easy of fusion. Among the purer varieties of clay, the most refractory are those in which the proportion of silica is greatest; and reaches its maximum in those substances which, although exhibiting many of the physical properties of clays, can scarcely be classified among them on account of their very large percentage of silica. Such, for example, are the different varieties of earth consisting of the siliceous remains of infusoria.

The composition of several varieties of British fire-clay is given on page 111.

The table of analyses on page 112 gives the composition of several foreign fire-clays.

LIME, MAGNESIA, ALUMINA, &c.—Lime, when heated alone, is one of the most refractory substances known, since at no temperature to which it has hitherto been subjected does it exhibit the slightest indication of fusion. Lime crucibles of small sizes are easily made, and may be sometimes useful when very high temperatures are required. The apparatus designed by Deville for the fusion of large quantities of

COMPOSITION OF BRITISH FIRE-CLAYS.

| | 1. | 2. | 3. | 4. | 5. | 6. | 7. | 8. | 9. | 10. | 11. | 12. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| $SiO_2$ . . . . . | 63·30 | 51·80 | 50·20 | 51·10 | 69·25 | 55·50 | 67·12 | 53·05 | 58·10 | 66·16 | 79·40 | 68·98 |
| $Al_2O_3$ . . . . . | 23·30 | 30·40 | 32·59 | 31·35 | 17·90 | 27·75 | 21·18 | 28·13 | 26·59 | 22·54 | 12·25 | 23·82 |
| $K_2O$ . . . . . | ... | ... | 2·32 | ... | ... | 2·19 | 2·02 | 4·19 | 1·21 | ... | ... | 0·07 |
| $Na_2O$ . . . . . | ... | trace | ... | ... | ... | 0·44 | ... | ... | ... | ... | ... | 0·49 |
| CaO . . . . . | 0·73 | ... | 0·36 | 1·46 | 1·30 | 0·67 | 0·32 | 0·17 | 0·40 | 1·42 | 0·50 | trace |
| MgO . . . . . | ... | 0·50 | 0·44 | 1·54 | | 0·75 | 0·84 | 1·20 | 0·99 | trace | ... | 0·17 |
| FeO . . . . . | 1·80 | 4·14 | ... | 4·63 | 2·97 | 2·01 | ... | ... | ... | 5·31 | ... | 0·10 |
| $Fe_2O_3$ . . . . . | ... | ... | 3·52 | | | ... | 1·85 | 2·48 | 2·97 | ... | 1·30 | 0·39 |
| $H_2O$, combined . | ... | ... | 9·69 | ... | ... | ... | 4·82 | 5·82 | 7·57 | 3·14 | 5·20 | 5·54 |
| $H_2O$, hygroscopic . | 10·30 | 13·11 | ... | 10·47 | 7·58 | 10·53 | 1·39 | 2·20 | 1·41 | ... | ... | 0·85 |
| Organic matter . | ... | ... | ... | ... | ... | | 0·90 | 2·82 | 1·21 | ... | ... | ... |
| | 99·43 | 99·95 | 99·12 | 100·55 | 99·00 | 99·84 | 100·44 | 100·06 | 100·45 | 98·57 | 98·65 | 100·41 |

1. Stourbridge, Worcestershire; used for glass-pots; by C. Tookey. 2. Brierly Hill, Staffordshire; by T. H. Henry. 3. Glascote, near Tamworth; by J. Spiller. 4, 5. Newcastle-on-Tyne; by T. Richardson. 6. Newcastle-on-Tyne; by Hugh Taylor. 7, 8, 9. Dowlais, South Wales; by E. Riley.—No. 7 is considered the best fire-clay of the district. 10. Glasgow; by J. Brown. 11. Ireland; by T. H. Henry; clay of excellent quality. 12. Lee Moor, Devon; by J. A. Phillips.

COMPOSITION OF FOREIGN FIRE-CLAYS.

| | 1. | 2. | 3. | 4. | 5. | 6. | 7. | 8. | 9. | 10. | 11. | 12. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| $SiO_2$, combined | 63·57 | 60·60 | 66·10 | 32·08 | 36·04 | 32·70 | 59·01 | 60·40 | 54·06 | 46·50 | 55·46 | 77·32 |
| $SiO_2$, as sand | | | | 25·04 | 21·04 | 26·40 | | | | | | |
| $Al_2O_3$ | 27·45 | 26 39 | 19·80 | 29·06 | 30·04 | 27·46 | 24·26 | 24·09 | 26·99 | 34·90 | 31·74 | 15·57 |
| CaO | 0 55 | 0·84 | ... | 0·04 | 0·56 | 0·10 | 1·32 | 0·55 | 0·85 | ... | 0·19 | trace |
| MgO | trace | ... | ... | 0·70 | 0·18 | 0·34 | 0·72 | 0·61 | 0·82 | ... | 0·14 | 0·13 |
| $K_2O$ | ... | ... | ... | 1·14 | 2·10 | 2·40 | 1·20 | 0·29 | 0·24 | ... | 2·49 | 0·67 |
| $Na_2O$ | ... | ... | ... | | | | | 0·22 | 0·33 | ... | 0·68 | 0·63 |
| $Fe_2O_3$ | 0·15 | 2·50 | 6·30 | 0·45 | 0·67 | 0·93 | 4 04 | 3·70 | 2·73 | 3·00 | 0·59 | 0·86 |
| $H_2O$, combined | 8·64 | 9·20 | 7·50 | 9·03 | 8·45 | 8·00 | 10·24 | 10·60 | 14·15 | 13·30 | 9 37 | 5·61 |
| $H_2O$, hygroscopic | 1·27 | ... | ... | ... | ... | ... | ... | ... | ... | ... | ... | ... |
| | 101·63 | 99·53 | 99·70 | 97·54 | 99·08 | 98·33 | 100·79 | 100·46 | 100·17 | 97·70 | 100·66 | 100·79 |

1. Beleu, Ardennes. 2. Dourdan, Seine-et-Oise. 3. Hayange, Moselle; by Salvétat.—No. 3 used for fire-bricks. 4, 5, 6. Clays from Belgium; by Bischof. 7. Schöningen, Hanover; by Streng. 8, 9. Kipfendorf, Saxe-Coburg; by Fresenius. 10. Almerode, Hessen-Cassel; by Berthier. 11. Vallendar, near Coblenz. 12. Mehlem, near Königswinter.

platinum, by the combustion of coal-gas with oxygen, consists of a box of cast-iron lined with rectangular blocks of lime.

Magnesia is quite as infusible as lime, and bricks of this material are sometimes employed. The mixture of lime and magnesia, obtained by subjecting dolomitic limestone to a very high temperature, is much used for lining the Bessemer converter in the basic process for steel-making.

Alumina is an extremely refractory substance, and cannot be melted at the highest temperature obtainable in our furnaces. According to Deville, crucible may be made of a mixture of hydrated or gelatinous alumina with a proper proportion of alumina which has been previously strongly heated. With two parts of gelatinous alumina, two of strongly heated alumina, and one of caustic lime, crucibles may be prepared which only soften slightly at the temperature of fused platinum. The mineral known as bauxite, which is essentially a hydrated silicate of aluminium, has sometimes been employed as a lining for furnaces.

Sesquioxide of chromium, $Cr_2O_3$, and chrome iron ore, $FeCr_2O_4$, are among the most refractory substances known. The former has been used as a bed or crucible for melting platinum in, while the latter has been applied in Russia as a lining for kilns for calcining dolomite, and for preventing contact between the siliceous brickwork and the hearth-lining in the basic open-hearth steel process, subsequently described.

FIRE-BRICKS.—The qualities required of fire-bricks differ in accordance with the purposes to which they are to be applied. Sometimes it is important that they should not become softened in any sensible degree by continued exposure to very great heat. In other cases it is necessary that they should withstand great and sudden changes of temperature. It is often essential that they should be capable of withstanding considerable pressure when strongly heated, and their capability of resisting the corrosive action of metallic oxides is also a consideration of importance.

It is seldom possible to produce a fire-brick capable of fulfilling all these conditions from clay alone, and, consequently, such additions of other materials are made as will afford bricks suited for the particular purpose for which they are required. Many varieties of clay which possess the requisite degree of infusibility are, when subjected to a rapid change of temperature, liable to split, from the expansion or contraction of the mass. These additions consist chiefly of pure siliceous sand, and ground and previously burnt fire-clay, which, without increasing the fusibility of the compound, have the property of rendering the material less liable to become disintegrated through the rapid application of heat or a sudden reduction of temperature.

The manufacture of refractory bricks is conducted in a very similar way to that of bricks employed for building purposes.

The fire-clay, after being for some time exposed to the air, is crushed under a pair of heavy edge-stones, where it is ground, either with or without the addition of silica, or of previously baked clay of the same description, until it has been reduced to the state of somewhat coarse powder. This falls through a hole in the bed, and is thence mounted by buckets on an endless chain into a large cylindrical sieve, by which it is divided into two classes. The coarser fragments, which do not pass through the meshes, are returned under the edge-runners, to be again ground, while the finer particles, which have passed through the apertures, are conducted by an endless belt to a convenient situation, where they are deposited under a small continuous stream of water. The mixture is subsequently incorporated in a pug-mill, and moulded into bricks in the way adopted for the more common varieties employed for building. A man and a boy can in this way, with a hand-mould, make and lay out to dry, 1,500 bricks as a day's work. When sufficiently dried, they are baked during several days in kilns containing from 15,000 to 20,000. A ton of coal is, on an average, required for the baking of every 3,000 bricks, which are all placed on edge, lengthways, and separated from one another by about a finger's breadth in order to allow a free passage between them of the heated gases produced by the combustion of the fuel, which is consumed at the extremity of the pile furthest removed from the chimney.

The following table gives the composition of fire-bricks from various localities :—

Composition of Fire-Bricks.

| | 1. | 2. | 3. | 4. | 5. | 6. |
|---|---|---|---|---|---|---|
| $SiO_2$ . . . . | 63·09 | 84·65 | 88·43 | 69·30 | 75·36 | 71·02 |
| $Al_2O_3$ . . . | 29·09 | 8·85 | 6·90 | 28·50 | 21·47 | 26·47 |
| $CaO$ . . . . | 0·42 | 1·90 | 3·40 | ... | trace | trace |
| $MgO$ . . . | 0·66 | 0·35 | trace | ... | ... | 0·44 |
| $FeO$ . . . . | ... | ... | ... | ... | 1·79 | trace |
| $Fe_2O_3$ . . . | 2·88 | 4·25 | 1·50 | 2·00 | ... | 0·80 |
| $K_2O$ . . . . | 1·92 | ... | ... | ... | ... | 0·50 |
| $Na_2O$ . . . | 0·31 | ... | ... | ... | 1·38 | 0·42 |
| $TiO_2$ . . . . | 2·21 | ... | ... | ... | ... | S. trace |
| | 100·58 | 100·00 | 100·23 | 99·80 | 100·00 | 99·65 |

1. Dowlais; by E. Riley. 2. From Windsor clay; a mixture of 30 per cent. of clay and 70 of sand; by Richardson. 3. From Pembroke; used in copper-works; by Napier. 4. Creuzot, France; used for blast-furnaces; by Berthier. 5. Lee Moor, Devonshire; by Abel. 6. From same locality; by J. A. Phillips.

The following analyses of an ordinary "blue fire-brick," from Buckley, North Wales, were made in the author's laboratory by Mr. W. T. Gent. Sp. gr. = 2·28 :—

| | 1. | 2. |
|---|---|---|
| $SiO_2$ | 72·45 | 72·63 |
| $FeO$ | trace | trace |
| $Fe_2O_3$ | 4·69 | 4·78 |
| $Mn_3O_4$ | ·30 | 35 |
| $Al_2O_3$ | 20·83 | 20·70 |
| $CaO$ | ·30 | ·29 |
| $MgO$ | trace | trace |
| $K_2O$ | ·61 | ·63 |
| $Na_2O$ | ·62 | ·57 |
| $H_2O$, hygroscopic | ·13 | ·16 |
| | 99·93 | 100·11 |

The "silica" fire-bricks of South Wales, which are probably the most infusible bricks employed in this country, consist almost entirely of silica, and, instead of being chiefly composed of fire-clay, are made from a disintegrated sandstone, found at Dinas and various other places in the Vale of Neath.

Two specimens of "Dinas clay," obtained from different mines, analysed by Mr. W. Weston in Dr. Percy's laboratory, afforded the following results:[1]—

| | 1. | 2. |
|---|---|---|
| $SiO_2$ | 98·31 | 96·73 |
| $Al_2O_3$ | 0·72 | 1·39 |
| $FeO$ | 0·18 | 0·48 |
| $CaO$ | 0·22 | 0·19 |
| $K_2O$ & $Na_2O$ | 0·14 | 0·20 |
| $H_2O$, combined | 0·35 | 0·50 |
| | 99·92 | 99·49 |

Ganister is a siliceous material, somewhat similar in composition to Dinas stone, found in the lower Coal-measures of South Yorkshire. It is to some extent plastic when ground and tempered with water, and is largely used for the lining of Bessemer converters and other purposes in the Sheffield steel-works.

CRUCIBLES.—Crucibles are commonly manufactured either by working the prepared clay on a potter's wheel, similar to that employed in making ordinary pottery, or by compressing it in moulds, which thus communicate to the mass the required form. Sometimes, also, although more rarely, they are prepared by covering with clay a mandrel made either of metal or of hard wood, and having the exact form and dimensions of the internal cavity of the vessel required. Crucibles should be capable of resisting sudden changes of temperature without fracture. They should also be nearly infusible, be unacted on by the ashes of the fuel by which they may be surrounded, and withstand the corrosive action and permeation of such substances as melted lead oxide. It would, however, be difficult to prepare crucibles capable of fulfilling all these conditions, and it is consequently found better to select the mixture in accordance with the use to which it is to be applied, than to attempt the manufacture of pots applicable to every purpose.

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[1] Percy's 'Metallurgy;' Fuel, p. 147.

When it is desired to prepare crucibles capable of withstanding sudden changes of temperature, the prepared clay is intimately mixed with various infusible bodies which impart to the mass the property of neither expanding nor contracting in a sensible degree on being strongly heated and afterwards rapidly cooled. These substances generally consist of siliceous sand, ground flints, calcined clay, graphite, or powdered coke.

The most infusible crucibles are prepared from clays containing the largest proportion of silica, and in which the amount of lime and oxide of iron is small. The infusibility of clay, like its power of sustaining sudden changes of temperature, may be much increased by a judicious admixture of materials, which, forming a kind of infusible ground-work, prevent the crucible from being deformed by exposure to a temperature by which it would otherwise be destroyed. The most efficient materials for this purpose are siliceous sand, graphite, or powdered coke, added to the clay in the proportion of about one-fifth; if, in the case of the two latter, a larger amount were used, although the infusibility of the crucible might be increased, the carbonaceous matter would be liable to become consumed and the crucible gradually destroyed.

The composition of several varieties of manufactured crucibles has been examined by Berthier, some of whose results are arranged in the following table:—

| Place of Manufacture. | $SiO_2$. | $Al_2O_3$. | $Fe_2O_3$. | MgO. |
|---|---|---|---|---|
| Crucibles, Hessian . . . . . . | 70·9 | 24·8 | 3·8 | ... |
| „ from Paris . . . . . | 64 6 | 34·4 | 1·0 | ... |
| „ „ Savignies, near Beauvais . | 72·3 | 19·5 | 3·9 | ... |
| „ „ England, for melting steel . | 71·0 | 23·0 | 4·0 | ... |
| „ „ St. Etienne, for do. . . | 65·2 | 25·0 | 7·2 | ... |
| Glass Pots „ Nemours . . . . | 67·4 | 32·0 | 0·8 | ... |
| „ „ Bohemia . . . . | 68·0 | 29·0 | 2·2 | 0·5 |

In order that an earthen crucible may be but slightly attacked by the bodies fused in it, it is necessary that the particles of which it is composed should be finely divided and closely compressed, and also that the materials of which it is made should not readily form fusible compounds with the substance operated upon.

The metals and their ores, with the exception of their oxides, generally exert little action on crucibles made of ordinary fire-clay, although galena, together with certain other substances, has the property of filtering through the pores of some earthen crucibles without exercising on their constituents any extensive chemical action. The degree of facility with which clay pots yield to the action of metallic oxides is usually tested by the fusion of litharge, which is maintained in the fluid state until the pot becomes pierced by its corroding action, when the time necessary to produce this effect is noted and compared with similar results obtained with other crucibles. Black-lead crucibles, and those in which ground coke has been incorporated, are attacked by fusible metallic oxides, through the gradual oxidation of the carbon, which results in the reduction of a portion of the metal.

The degree of fusibility of crucibles and other refractory bodies may be ascertained by a direct experiment conducted in the following manner:—A piece of the substance to be examined, which for this purpose should present numerous sharp edges, is heated, in a refractory crucible, lined with powdered charcoal, to the fullest extent possible in a large wind-furnace. The pot and its contents are then allowed to cool, and on afterwards examining the contents, it will be observed whether the thin edges of the fragment have become rounded or have been rendered translucent; in which case it affords a sufficient indication that a commencement of fusion has taken place.

The permeability of crucibles by liquids may be determined by filling them with water, and noting what time elapses with each variety before any appearance of dampness is perceived on the outside.

For the purpose of ascertaining their power of resisting sudden changes of temperature, crucibles may be thrown, without any previous annealing, into an intensely heated furnace, and afterwards withdrawn and at once exposed to a current of cold air.

Crucibles are used both in the unburnt and burnt state. Small crucibles are usually kiln-burnt before they are used, but the large Stourbridge clay casting-pots, which are extensively employed by brass-founders, are never previously burnt. They are first gradually and thoroughly dried by the maker in properly constructed stoves, and are afterwards kept for use by the founder on shelves in some dry and warm situation in the casting-shop. A fire is made in a cold furnace and covered to a depth of a few inches with coke, broken to a convenient size. On this the crucible is placed in an inverted position, and the furnace filled up with coke. When the crucible has become uniformly red hot it is withdrawn, and immediately replaced with its mouth upwards. It is a somewhat remarkable fact that pots of this description which are at first put into a furnace with the mouth upwards almost invariably crack.

Four different kinds of crucibles are used by assayers in this country, viz., the London, the Cornish, the Hessian, and the French; of these, the two former are most extensively employed.

London crucibles are of a reddish-brown colour, and are close in grain, but are liable to crack, and consequently require to be very gradually heated. They resist the corrosive action of fused litharge remarkably well, and, with careful management, are very serviceable for fusions with oxide of lead. The Patent Plumbago Crucible Company, of Battersea, have for some years manufactured crucibles closely resembling in appearance the "creusets de Paris," but they are considerably thicker.

Crucibles are manufactured in Cornwall for the use of copper-assayers. They are usually made round, and of two sizes, one of which fits into the other; those of the larger size are 3 inches in diameter at top, and 3½ inches in height, outside measure. These crucibles are not capable of withstanding very high temperatures, or of resisting, for any considerable time, the action of melted litharge. They can, however, be introduced into a hot furnace without cracking, and can be more generally employed for assays than almost any other description of crucible. They were formerly made by Mitchell of Truro and by Juleff of Redruth, and those of the two makers were, for general purposes, almost equally good, although Juleff's crucibles, which were made of a mixture of Teignmouth and Poole clay and of sand from St. Agnes Beacon, were somewhat less rapidly acted on by fused oxide of lead. Mr. Juleff has transferred his business to the Battersea Crucible Company, but Mr. Mitchell continues to make pots of good quality.

The shape of the Cornish pots is somewhat inconvenient from their flatness at the bottom, but in all cases where sudden changes of temperature are to be undergone, they are to be preferred to every other variety, except those manufactured by Beaufay of Paris.

Hessian crucibles are sold in nests of six, gradually diminishing in size so as to successively fit into one another. They are made of a mixture of Almerode clay and sand, and are generally triangular at top, so that their contents may be conveniently poured from either of the corners. They withstand a tolerably high heat without softening, but are liable to be cracked by sudden changes of temperature, and are readily permeated by fused metallic oxides.

French crucibles are circular and considerably deeper in proportion to their width than the Hessian. They are also made of more finely ground materials, and present a smooth surface both inside and out. Those made by Beaufay are of excellent quality, and not only withstand a high temperature, but likewise retain melted litharge for a long time without becoming pierced. They are, however, somewhat brittle, and require that the tongs used for withdrawing them from the fire should not grasp them too roughly. They are by far the best crucibles for assays of gold and silver ores, in making which large quantities of litharge are employed.

Plumbago or black-lead crucibles are more frequently employed as melting-pots for the fusion of metals and alloys than for the purposes of assaying, and are manufactured of good quality by the Patent Plumbago Crucible Company, and others.

## IRON.

Iron, as produced on the large scale, invariably contains in combination some proportion of various foreign elements, which, although small, may have a very marked influence upon its strength and working qualities. In a systematic investigation of its properties, it is therefore necessary to obtain the metal as nearly as may be in a chemically pure condition, for which purpose three different methods are adopted, namely—

1. Fusion of wrought-iron with iron oxides.
2. Reduction of iron oxides by hydrogen.
3. Electrolysis of a ferrous salt.

1. The first of these methods is applicable to the purer kinds of wrought-iron, such as the wire used for pianos, which contains only a small proportion of carbon. This, when intimately mixed with about one-tenth of its weight of iron oxide and strongly heated, loses its carbon, which is oxidized to carbonic oxide, a corresponding amount of iron being reduced. The best way of obtaining sufficient contact between metal and oxide is to cut the wire into small pieces, and to partially oxidize it by heating it in the air, or, what is better, exposing it to a current of steam in a porcelain tube at a red heat. The superficially oxidized metal is then melted in a clay crucible under a glass flux free from heavy metallic oxides, which requires about an hour of full white heat in an air-furnace. Combined carbon and silicon are removed by the oxygen of the ferric oxide, and the excess of the latter fluxed with silica from the crucible forms a slag, while the metal collects as a well-melted button at the bottom of the crucible.

2. Pure ferric oxide, when heated to redness in a current of pure dry hydrogen, is completely reduced with an equivalent production of water, thus,—

$$Fe_2O_3 + 6H = 2Fe + 3H_2O.$$

The metal obtained is a dull dark-coloured powder, which should be uniformly grey if the reduction is complete, otherwise it is black from intermixed magnetic oxide. For complete reduction, steady heating and a rapid current of gas to remove the water-vapour as it is formed are essential. This is the most exact method of obtaining iron from its compounds, but has the disadvantage of giving a form of metal which is finely divided and extremely susceptible of oxidation; especially if produced at a low temperature, when it is pyrophoric, and burns spontaneously on exposure to the air at ordinary atmospheric temperatures. When the reduction is effected at a strong red heat, the metal is less susceptible of oxidation. Its pyrophoric tendency is also enhanced by the presence of foreign substances, such as unreduced iron oxide, alumina, &c.

The following modification of the hydrogen method of reduction was employed by Matthiessen. Ferrous and sodium sulphates mixed in nearly equal proportions are melted in a platinum crucible, and heated until no more sulphur dioxide is given off. This gives a crystalline ferric oxide, which can be more easily purified from adherent foreign matters than the hydrated oxide precipitated from solution by alkalies. The sodium salt is removed by long and careful washing with boiling water, which is continued as long as traces of sulphuric acid can be found in the washings. The oxide is then reduced by hydrogen, and the reduced metal, when cooled and solidified by compression in a steel mould, is melted in a lime crucible by the oxy-hydrogen flame. Iron so obtained is free from all impurities except a minute trace of sulphur.

Iron may be obtained in cubic crystals when the vapour of ferrous or ferric chloride is heated in hydrogen. This requires a very high temperature, the reduction being effected in a porcelain tube.

3. Iron may be obtained in coherent films by the electrolysis of solutions containing ferrous and magnesium sulphates, provided that they be kept neutral by the addition of carbonate of magnesium. The deposit, at first brilliantly metallic, subsequently becomes velvety in lustre and covered with gas-bubbles. It is hard (between 5 and 6 of the mineralogical scale) and brittle, but when heated to redness *in vacuo* acquires softness and tenacity, and takes a darker colour. The metal obtained by the electrolysis of a mixture of ferrous chloride with sal-ammoniac forms brittle warty masses, which are hard enough to scratch glass. This hardness is due to the absorption of hydrogen, which is given off when the metal is heated *in vacuo* to the extent of from 240 to 250 times its volume, corresponding approximately to a composition of $Fe_{13}H_2$. In this respect iron resembles palladium, but, unlike the latter metal, when once deprived of its hydrogen it does not re-absorb it when used as the negative electrode in a voltameter.

According to Crookes, metallic magnesium decomposes a solution of a ferrous or ferric salt, with evolution of hydrogen and the separation of pulverulent metallic iron. The metal so obtained is exceedingly pure, dissolving in acids without residue, and evolving hydrogen free from odour.

CRYSTALLINE FORM.—Iron crystallizes in the cubic system, the form most generally observed being cubes or octahedra, the latter often in hollow-faced or skeleton shapes, and grouped into tree-like aggregates. Solid masses obtained by fusion, that have not been subjected to hammering, when polished and etched show a granular crystalline surface, often with interlacing plates, forming a square or triangular network similar to those obtained in meteorites, and known as Widmannstätten's figures.

SPECIFIC GRAVITY.—The maximum density of electrolytic iron appears to be about 7·9. Most commercial varieties of the metal are considerably lighter, or from 7·5 to 7·7. It may be diminished by strains, such as are developed by cold-rolling or wire-drawing; and in such cases the higher density is restored by annealing.

TENACITY.—Iron is one of the strongest of the malleable metals;

the purest commercial kinds, *i.e.*, those containing least carbon, bear a tensile strain of 14 tons per square inch without permanent elongation, and break at a strain of about 22½ tons. The strength of steel in the form of wire of 0·093 inch section, when hard, has been found to be as much as 154 tons per square inch, and its elastic limit 53 tons per square inch.

Thermal Relations.—Pure iron melts at a temperature which has been estimated at about 1,500° to 1,600° C., being more fusible than the allied metals manganese, cobalt, and chromium, but less so than nickel. At the highest artificial temperatures, such as those of the electric arc and the Bessemer process, it may be volatilized, and burns with the production of a dense orange-red smoke.

Optical Characters.—When polished, iron has a characteristic dark grey colour, being less white than platinum, and not so blue as lead, with a perfect metallic lustre. When obtained in very thin films by electrolytic deposit on platinized glass (from $\frac{1}{500.000}$ to $\frac{1}{350.000}$ of an inch thick), such as have been used by Kundt in experiments on the electro-magnetic rotation of the plane of polarization, it transmits a pale yellowish or brownish light.

Electrical Relations.—The electric conductivity of soft iron is about 16 per cent. of that of copper or silver at ordinary temperatures, but the presence of even minute quantities of foreign matters lowers it very considerably, manganese being the most active in this respect. According to Preece, wires of equal resistance (one ohm) made of 99·7 and 99·11 per cent. iron, weigh respectively 4,500 lbs. and 5,974 lbs. per mile, whence it appears that diminishing the iron by $\frac{1}{150}$th increases the resistance by one-third. The conductivity is also diminished by hard-drawing and increased by annealing.

Soft iron becomes magnetic when brought into contact with, or into the field of, a permanent or electro-magnet, but such magnetism is only temporary. The harder varieties containing carbon, though of lower magnetic capacity, become permanently magnetized in virtue of a property known as *coercive force*, which is most developed in the very hard alloy containing tungsten known as tungsten steel. Manganese diminishes the magnetic capacity of iron very rapidly; the alloy with 25 per cent. of manganese is not attracted by the magnet.

The following determinations of the magnetic capacity of different kinds of iron have been made with Professor Hughes' induction balance:—

| | Electro-Magnetic Capacity. | Residual Magnetism. |
|---|---|---|
| Hadfield's manganese steel . . . . | 2 | 0 |
| Cast-iron, hardened . . . . . | 160 | 10 |
| Shear-steel „ . . . . . | 292 | 47 |
| Cast-iron, annealed . . . . . | 406 | 42 |
| Huntsman's cast-steel, annealed . . | 450 | 45 |
| Shear-steel „ . . . . | 472 | 40 |
| Charcoal iron „ . . . . | 577 | 11 |
| Best charcoal iron „ . . . . | 610 | 5 |

Electro-deposited iron when saturated with hydrogen is highly magnetic, and a thin strip of such iron, when suspended, comes to rest in the plane of the magnetic meridian, like a compass needle. The transparent films previously noticed are found by Kundt to be extremely energetic in rotating the plane of a polarized ray, the rotatory power, for equal thicknesses, being computed to be 34,000 times that of glass, and 1,462 times that of quartz.

**Relation of Iron to Oxygen.**—Pure iron may be preserved indefinitely without alteration in dry air or in water that is free from air and carbon dioxide, at ordinary atmospheric temperatures; but when heated to redness in air or oxygen, it burns with the production of magnetic oxide ($Fe_3O_4$). The same result is obtained when pyrophoric iron ignites spontaneously. When steam is passed over iron at a red heat, it is decomposed with the evolution of hydrogen and a similar production of magnetic oxide. This reaction goes on continuously as long as the hydrogen is removed, but if it is allowed to accumulate in the atmosphere, the decomposition is stopped as soon as a particular gas-tension, which is constant for any given temperature, is reached. A similar result is observed in the inverse operation of the reduction of magnetic oxide by hydrogen, which is stopped as soon as the water-vapour has attained a particular tension in the hydrogen atmosphere. The same class of phenomenais observed in the reduction of an iron oxide by carbonic oxide, and has an important bearing upon the reduction of the ore in the smelting process.

Iron combines with oxygen in several proportions, but of these only three are of consequence metallurgically considered, namely, protoxide or ferrous oxide, peroxide or ferric oxide, and their compound known as magnetic oxide of iron.

**Ferrous Oxide.**—Ferrous oxide or protoxide of iron having the composition FeO is known only in combination. It may be obtained as a hydrate ($FeO{\cdot}H_2O$) by the addition of an alkali to the neutral solution of a ferrous salt, when it goes down as a white precipitate, which, however, rapidly changes colour, becoming brown by the absorption of oxygen from the air, and changing to a hydrate of the sesquioxide. The same instability characterizes all the salts of ferrous oxide, such as the sulphate and carbonate, and also the chloride, all of which pass into the ferric state, unless protected from the action of the air.

**Ferric Oxide.**—Peroxide, or sesquioxide of iron has the composition $Fe_2O_3$ (containing iron 70, oxygen 30 per cent.), and occurs in nature in several states of aggregation, which are distinguished by particular names, as described under "Iron Ores." It may be prepared artificially by exposing metallic iron, at or near red heat, to the continued action of air, when it is obtained as a dense red powder, varying in colour from orange red to purplish red, according to the state of aggregation. Another and more general method of preparing it is by the calcination of ferrous sulphate, $FeSO_4$, which is decomposed with the production of ferric oxide, $Fe_2O_3$, and the anhydrous sulphuric acid, $SO_3$. The oxide so obtained is a red powder known as "crocus," which, when carefully levigated in water, gives powders largely used for polishing under the name of "rouge."

When steam is passed over ferric chloride ($Fe_2Cl_6$) at a red heat, hydrochloric acid and ferric oxide are formed. The latter is in brilliant crystalline scales of a bluish grey colour and strong metallic lustre, exactly similar to the natural mineral known as oligsite. Specular iron ore occurs as a common constituent of volcanic rocks.

Ferric oxide, however prepared, when pulverized gives a bright red powder of about sp. gr. 5, which is, under ordinary conditions, stable in the air; but when brought to a high temperature, such as that of the oxy-hydrogen jet, it is partially decomposed with the production of magnetic oxide ($Fe_3O_4$). It is soluble in oxygen acids, forming salts which have a tendency to desiccate into complex basic compounds, and are comparatively unstable. With hydrochloric acid it gives ferric chloride ($Fe_2Cl_6$), which is very soluble in water.

When the solution of a ferric salt is treated with an alkali, a chalky flocculent precipitate of a yellowish brown colour is produced, which is a hydrate of ferric oxide or a ferric hydrate ($Fe_2O_3 \cdot nH_2O$). The proportion of water varies with the temperature at which the separation takes place; the most usual variety being the sesquihydrate ($2Fe_2O_3 \cdot 3H_2O$), which is of the same composition as the natural mineral, brown iron ore, or limonite. By long-continued boiling a deeper coloured substance, the monohydrate ($Fe_2O_3 \cdot H_2O$), is obtained. This is of the same composition as the natural mineral Göthite.

RUST OXIDE.—Metallic iron is not subject to change at ordinary temperatures in perfectly pure air, or in water free from air and carbon dioxide; but when exposed either to moist air or fully aerated water it is rapidly attacked with the formation of a dull brown crust, more or less coherent, of so-called *rust*, which is essentially a hydrated ferric oxide. The change is accompanied with the decomposition of water, and incidentally a small quantity of ammonia is formed by the combination of nascent hydrogen with the nitrogen of the air, and is absorbed by the rust. The film of oxide produced being strongly electro-negative to the metal below, a voltaic action of some energy is set up under the influence of water in the air, and continues until the metal is entirely oxidized or converted into rust. This change is accompanied with a great increase of volume, the bulk of the rust formed being about ten or twelve times that of the original metal, and is capable of producing considerable mechanical action, bars of iron being often burst asunder when rusting is set up by infiltration of water through surface cracks.

MAGNETIC OXIDE.—When metallic iron is exposed to a very high temperature in air or oxygen, it burns with the production of black molten globules that are attractable by a magnet, having a composition represented by $Fe_3O_4$ or $FeO \cdot Fe_2O_3$, containing iron 72·4, oxygen 27·6 per cent. The oxide that forms on the surface of masses of wrought-iron at the heat of working in the forge or rolling-mill, known as scale-oxide, is of a similar composition as a whole. The calcination of ferrous carbonate also gives rise to magnetic oxide, the expulsion of carbon dioxide being accompanied by the absorption of oxygen, owing to the instability of ferrous oxide in the free state.

Magnetic oxide is one of the most stable compounds of iron, both at high temperatures and also as regards resistance to atmospheric alteration, both of which properties have important bearings on metallurgical practice.

IRON AND SULPHUR.—When finely divided iron and sulphur are heated together, combination takes place with a considerable development of heat; the product, normal ferrous sulphide, or iron monosulphide, $FeS$, containing iron 63·6, sulphur 36·4 per cent. It is also found when hydrogen sulphide or an alkaline sulphide is added to the solution of a ferrous salt not containing free acid. The sulphide obtained by the first method is fusible, giving a dark-coloured substance of a semi-metallic lustre and brassy iridescence, crystallizing in the cubic system, which is fairly stable at high temperatures in a neutral atmosphere, but when exposed to air at a low red heat (as in a calcining furnace), is converted into sulphur dioxide and ferric oxide. When heated to redness in steam, magnetic oxide is produced. The finely divided precipitated sulphide is also very unstable, being rapidly changed by exposure at ordinary temperatures to ferrous sulphate.

Iron monosulphide occurs in nature as an independent mineral called troilite, which is, however, extremely rare, being confined to meteorites; but in combination with the similarly constituted sulphides of other metals it appears in numerous species, such as Eisennickelkies ($2FeS{\cdot}NiS$), copper pyrites ($CuFe{\cdot}S_2$) and black zinc blende ($nZnS.FeS$), while similar substances are commonly formed artificially in so-called matte or regulus.

Finely divided or spongy iron, obtained by reduction in hydrogen or carbonic oxide, when heated in furnace gases containing sulphur dioxide, decomposes it, taking up the sulphur.

Bisulphide of iron, ferric persulphide ($FeS_2$) iron 46·7, sulphur 53·3 per cent., is the most important natural sulphide of iron. It occurs in two forms, cubic or yellow pyrites, and white rhombic pyrites or marcasite. Substances of similar composition may be prepared artificially by heating the lower sulphide in ferric oxide with hydrogen sulphide, alkaline bisulphides or sulphur, at a carefully regulated heat below redness, and above 100° C. But these methods are of little practical interest. In nature it is commonly formed by the reducing action of decomposing organic matters upon ferrous sulphate, and of organic sulphur compounds upon ferric hydrate. The crystallized varieties differ considerably in their resistance to atmospheric alteration, the yellow or cubic forms, when in compact masses and large crystals, may be kept unchanged for long periods, but when finely divided and exposed to damp, both components are oxidized, with the production of hydrated ferrous sulphate, ($FeSO_4.7H_2O$), and sulphuric acid. This is the action known as vitriolization, which is one of the most important agents of change in the vicinity of mineral deposits, such as the formation of alum, gypsum, the various sulphates of copper, &c., by the reaction of sulphates and sulphuric acid upon the soluble or decomposable constituents of the adjacent rock.

White iron pyrites is more readily vitriolized than the cubic varieties, and is a prominent cause of the spontaneous ignition in coals, which often contain it in a finely divided state.

The ultimate product of the atmospheric alteration of pyrites is ferric hydrate (limonite or brown iron), a change which is illustrated on the largest possible scale in various mineral districts. The sulphur in such cases is probably removed as a soluble sulphate of some other metal, as the whole of the iron of the original pyrites may be found in the ferric hydrate produced.

When iron pyrites is heated to redness in closed vessels, the whole or a portion of the second atom of the sulphur may be volatilized and sublimed as such, leaving a residue of ferrous sulphide or of magnetic sulphides; but when the heating takes place with access of air, it burns with the ultimate production of sulphur dioxide and ferric oxide. This property is taken advantage of in the use of pyrites for making sulphuric acid, the residue forming a coherent mass of the same shape as the original pyrites. In order to obtain complete desulphurization it is necessary to prevent fusion, in which case ferrous or magnetic sulphides would result.

When pyrites is finely interspersed through ferrous carbonate, it is mainly converted by calcination into ferrous sulphate, which can only be removed from the ore by long-continued exposure to the air, by solution in rain, or by artificial watering.

Sulphide of iron also occurs in nature under the name of pyrrhotine or magnetic pyrites. It crystallizes in hexagonal forms, is slightly magnetic, and is chiefly valuable for containing nickel and cobalt in Sweden, and gold in Brazil and some other countries.

IRON AND CARBON.—When ferric oxide is heated to redness in a current of carbon dioxide or any hydrocarbon gas, such as olefiant gas, $C_2H_4$, marsh gas, $CH_4$, or the vapour of benzole, paraffin, or analogous substances, it is reduced in the same way as by hydrogen, but with this difference, that carbon is invariably separated and taken up by the metal. This may be rendered apparent by dissolving the reduced metal in dilute sulphuric acid, when the hydrogen given off has a fetid smell resembling that of crude petroleum, which is not the case when the reduction has been effected by hydrogen. The same result is obtained when pure iron is heated with carbon in any form, whether as charcoal, graphite, or diamond, the amount taken up depending chiefly upon the temperature at which the operation is performed and its duration; the maximum being for theoretically pure metal about 4 per cent. This change, known as carburization, is the most important fact in determining the nature and properties of commercial iron, which are mainly classified by their proportion of contained carbon. Thus—

1. Malleable or wrought iron contains less than 0·2 per cent. of carbon, is infusible except in special furnaces, gives unsound castings, and cannot be tempered or hardened by quenching in water from a red heat; but can be welded and generally wrought under the hammer, being eminently malleable and ductile.

2. Steel includes all kinds of iron containing not more than 1·75 per cent. of carbon. These vary considerably in fusibility, hardness, susceptibility to tempering, and malleability, the first two properties being increased by increase of carbon, while the others are diminished.

The limit of about 0·4 or 0·5 per cent. may be taken as that of ordinary malleable steel, above which lie the more special qualities of tool and cutlery steel, which are chiefly valuable for their hardness. A metal with about 2 per cent., which cannot be forged but is slightly malleable in the sense of bearing a slight hammering without breaking, is used for draw-plates.

3. Cast-iron includes all metal with more than 2 per cent. of carbon up to a maximum of about 4 per cent. This, as the name implies, finds its most important application in the production of articles cast to patterns in moulds.

CONDITION OF CARBON IN IRON.—Carbon may exist in cast-iron and steel, either in chemical combination or interspersed through the mass of the metal in the crystalline form of graphite. When it is in the former condition, the metal is hard, somewhat largely crystalline, uniform in character, and silvery white in colour—this is known as white-iron; when it is in the latter the structure is not uniform, the brilliant particles of metal being mixed with dark scales of graphite, giving a general grey tint, the depth of which increases with the proportion of the carbon so separated—this is known as grey-iron. When white cast-iron is dissolved in dilute sulphuric acid, a considerable proportion of the carbon being set free in the presence of nascent hydrogen combines with the latter gas, forming hydrocarbons, which are recognised by the strong smell, or even, in some cases, may be obtained as condensable oils. Grey-iron, on the other hand, dissolves up, leaving the bulk of its carbon behind in the form of graphitic scales, which are not affected by acids. If, however, white-iron is brought into solution without the decomposition of water, as, for instance, by cupric chloride, which is decomposed with the production of ferrous chloride and metallic copper, the carbon separates as an amorphous sooty powder, while the graphite remains with its characteristic crystalline character. This method is of wide application in the analysis of iron and steel. Apart from the influence of other elements which will be subsequently considered, the condition of carbon in cast-iron is largely determined by the method adopted in casting; the same metal which is grey when cast in sand moulds and slowly cooled, will be white when cast in thick iron moulds, which allow of rapid cooling, or, as it is termed, *chilling*. In the first case the carbon is graphitic, while in the latter it is in the combined condition.

In those varieties of iron that contain less carbon than cast-iron the same general principle holds. Thus steel when hardened by quenching in water from a red heat dissolves without residue, but when brought back to the soft state by annealing, a flocculent residue is obtained. This consists mainly of carbon, but not entirely, as it contains some iron, corresponding approximately to the composition $Fe_4C$, and is known as *semi-combined* or *hardening carbon*.

IRON AND SILICON.—When ferric oxide intimately mixed with finely divided quartz or sand and charcoal is in a crucible exposed to the highest attainable temperature of an air-furnace, the metal obtained contains both carbon and silicon, and the proportion of the latter may, under favourable conditions, amount to 20 per cent. Unlike carbon, however, silicon appears to be unable to exist in the pure or graphitic state, being invariably in combination. These compounds, which resemble cast-iron, but are harder and more brilliant, were formerly curiosities, but are now made for special purposes of steel manufacture under the name of *silicite* or *ferro-silicon*. The latter name is not strictly correct, as will be subsequently shown, manganese being invariably present in considerable quantity as well as iron.

Silicon is also a common constituent of ordinary cast-iron up to 1·5 or 2 per cent. in those varieties used by the iron-founder; the proportion being largest in the darker coloured varieties. The large-grained, dark grey metal used in the Bessemer process, as originally constituted, contains about 2½ per cent. When containing 5 to 7 per cent., the metal is known as *glazy-* or *glazed-iron*. This is only an accidental product, and has no direct use. Mottled- and white-irons, such as are used for conversion into malleable iron, contain less silicon, and should, if possible, be entirely free from it, although this latter condition can only be attained by certain special methods of manufacture.

The compounds of silicon and iron, although stable at high temperatures when melted in a reducing atmosphere, are easily decomposed when heated in an oxidizing atmosphere, both iron and silicon being oxidized, with the formation of normal or dibasic ferrous silicate, $Fe_2SiO_4$. This substance, which crystallizes in forms analogous and almost identical with those of the mineral olivine, a similarly constituted ferrous magnesium silicate, may be easily formed by melting quartz, sand, and ferric oxide mixed in the proper proportions in an iron crucible with sufficient carbon to reduce the latter to ferrous oxide at a strong red heat. The product, which runs very liquid, is a nearly black crystalline regulus with a semi-metallic lustre, which in the thinner plates is sometimes of an olive-green tint. It is usually opaque and somewhat magnetic, which is, however, due to the inclusion of magnetic oxide of iron. The slags of all processes of iron-smelting in which oxidation is involved, such as the direct production of the malleable metal from the ore and the fining or conversion of cast-iron into malleable iron, consist wholly or in part of ferrous silicates, and as their production involves the formation of two molecules of ferrous oxide for each molecule of silica, or for every unit of silicon 3½ times its weight of iron is scorified and rendered useless, it will be easily seen that the exclusion of silicon as much as possible from all metal destined for forge purposes is essential to economic working. It is different, however, with foundry and steel-making metal, where, as will be subsequently shown, silicon has valuable properties. Ferrous silicate may be partially reduced by heating with carbon, when two-thirds of the metal are separated, the remainder forming a silicate of the constitution $Fe_2Si_3O_8$ (diferrous trisilicate); but by the substitution of another base, such as lime, complete reduction may be effected. This latter condition is utilized in the treatment of the large quantities of slag or cinder produced in puddling- and re-heating furnaces, which contain more than half their weight of iron, and are largely used in admixture with ores in the blast-furnace. Pig metal so produced is known as *cinder-pig*, in contradistinction to that smelted from freshly mined ores, or *mine-pig.*

When ferrous silicate is heated in air at a temperature below its melting-point, it is wholly or partially changed by oxidation into a mixture of ferric oxide and silica, which substances, being chemically indifferent, form a material which is exceedingly refractory, so long as it is not subjected to the action of reducing gases. This property has been utilized for the production of a material for puddling-furnace bottoms, which is obtained by calcining tap-cinder broken into lumps in kilns, when a certain proportion liquates out as a fusible slag, leaving a refractory residue known as *bull-dog.*

IRON AND PHOSPHORUS.—These elements enter very readily into combination, forming several compounds. Among the best-defined are the following:—

| | Phosphorus. | Iron. | Sp. Gr. |
|---|---|---|---|
| $Fe_3P_2$ . . . . | 57·53 | 42·47 | 5·04 |
| $Fe_2P$ . . . . | 42·47 | 57·53 | 5·21 |
| $Fe_4P$ . . . . | 21·68 | 78·32 | 5·74 |

The first of these is formed by heating iron, reduced by hydrogen, in phosphorus vapour, and the last, which is the most stable, by partially reducing ferrous phosphate with carbon, under a covering of salt, in a crucible at a strong heat. The amount of carbon must be so proportioned as to prevent the separation of metallic iron. The phosphates of earthy bases, when heated in contact with an oxide of iron and carbon, also give rise to iron phosphides, which are diffused through the mass of the reduced iron. This condition obtains in iron-smelting; practically the whole of the phosphorus contained in the ore, flux, and fuel passing into the reduced metal. Phosphorus so combined is exceedingly stable, and can only be completely removed from malleable iron or steel by special methods, involving the use of high temperatures and large quantities of fluxes.

The effect of phosphorus upon malleable iron is essentially one of hardening, raising its ultimate tensile strength, but diminishing its extensibility or working property. Up to about 0·3 per cent., the working quality of the metal is not affected, but beyond that limit cold-shortness, or a tendency to break when hammered cold, is set up. This is very marked with 0·8 per cent., and with 1·0 per cent. the metal becomes brittle. In steel the injurious effect is more marked, 0·1 per cent. being about the permissible limit. Phosphorus in cast-iron increases the fluidity of the metal, and for this reason that smelted from bog ores is of value in the production of small and intricate ornamental castings, although it is unfit for castings requiring great strength.

Iron and Nitrogen.—When iron wire is heated for several hours in a current of dry gaseous ammonia, the metal becomes somewhat whiter in colour and harder, and there is an increase in weight. The amount of the latter change is variously stated by different observers at from 0·2 to 12 or 13 per cent., and appears to depend upon the amount of surface exposed. Nitrogen has also been found in very small quantities by Bouis and Boussingault (0·005 to 0·124 per cent.) in commercial iron and steel. Allen has also demonstrated its existence in steel ingots.

It is doubtful, however, whether this constituent is of any importance in modifying the structural properties of the metal. In the method of steel-conversion by cementation, the presence of cyanides has a marked effect in accelerating the process.

Iron and Manganese, &c.—From the metallurgical point of view, these metals are very closely connected; in fact, the metallurgy of the latter can scarcely be said to exist apart from that of iron. Manganous oxide, MnO, unlike ferrous oxide, is not reduced by carbonic oxide, but requires the direct contact of carbon; neither is it possible practically to obtain the pure metal, the simultaneous reduction of iron, with which the manganese unites, being a condition for obtaining it on the large scale. Pig metals containing manganese, in which iron predominates, are known as manganiferous cast-iron and spiegeleisen; the latter name, signifying looking-glass-iron, refers to the large platy structure of the metal when broken. Spiegeleisen contains more combined carbon than white cast-iron, or up to as much as 6 per cent. The maximum amount of carbon taken up by manganese is said to be about 8 per cent. The most manganiferous varieties are known as ferro-manganese. These alloys have no direct use, but are essentials in the modern practice of steel-making on the large scale. The largely platy crystalline structure of spiegeleisen is not apparent when there is more than 25 to 30 per cent. of manganese present.

Manganese may exist in malleable iron, and in steel in small proportions, up to 0·4 or 0·5 per cent., without altering their properties, except somewhat increasing both tenacity and malleability. About 1·5 per cent. gives a very hard tool-steel, when the carbon is similarly increased. Latterly, alloys known as Hadfield's manganese steels, with from 7 to 30 per cent. of manganese, have been made. They are remarkable for combining extreme hardness and toughness.

Chromium combines with iron and carbon, forming chromiferous pig-irons, somewhat similar in appearance to those containing manganese. To steel it communicates great hardness without diminishing its toughness. Tungsten may be contained in steel up to a considerable percentage, communicating intense hardness and great coercive force for magnetism; it is doubtful, however, whether there is any true combination.

Copper and iron do not form any true alloy except in the presence of a third metal; in the cupriferous pig-iron, occasionally produced in blast-furnaces, the copper is interspersed through the mass, preserving its characteristic colour. To malleable iron and steel copper is said to communicate red-shortness, although this is not now generally accepted, the effect being attributed to the concurrent presence of sulphur. Arsenic and antimony are very prejudicial to iron, although their presence can only be considered as exceptional. The effect is similar to that of phosphorus.

TITANIUM may be present in pig-iron to the extent of about 1 per cent. when titaniferous ores form part of the smelting charge; but it is not known to exert any special effect upon the quality, and when such metal is converted into malleable iron or steel, the titanium is entirely eliminated.

VANADIUM has been found by Riley in the pig-iron of Wiltshire and Northamptonshire smelted from oolitic brown iron ores, and the same element has been found in the Cleveland ores and in pisolitic ores in Bavaria and Würtemberg. A minute quantity was also detected by Sefström in bar-iron made from the magnetic ore of Taberg in Sweden, which was in special repute for wire-drawing; but no special influence, beneficial or otherwise, can be attributed to its presence in the present state of knowledge of this subject.

ALLOYS OF IRON.—The combinations of iron with the metals proper are very few, and are mostly confined to those nearest to it in chemical characteristics. Zinc takes up about 7 per cent. of iron when kept melted for a long time in iron vessels, a condition which is realized in the process of galvanizing. The hard zinc so obtained is a source of loss in the process, as it has only received certain special applications in working bronzes or compound copper alloys. Tin and iron alloy freely, but their compounds are useless. A small proportion of tin acts very prejudicially upon malleable iron, a fact which interferes with the use of tin-plate waste for re-manufacture into bar-iron. Nickel alloys readily with iron without affecting its malleability, as is evidenced by the native irons of meteoric origin, which are as a rule nickeliferous. Neither lead nor silver unite with iron when melted, a property which is largely utilized by the lead-smelter and refiner; gold is readily taken up when the two metals are melted together. Platinum and iron alloys occur native, and are easily formed, combination taking place at temperatures far below the melting-point of either metal. This fact is often brought into disagreeable prominence when iron compounds are excessively heated in platinum crucibles. According to Faraday and Stodart, 1 per cent. of platinum in steel gives a very tough and fine-grained product, and similar results are obtained with other metals of the platinum group, such as palladium, rhodium, osmium, and iridium

## IRON ORES.

Iron ore is most abundantly distributed over the earth's surface, both as native metal and as a component, either principal or subordinate, in a great variety of complex minerals. For metallurgical purposes the latter, and among these only the oxides and carbonates, are of primary importance.

NATIVE IRON.—Solid bodies have often been observed to fall through the atmosphere, and these, when picked up, are found in many cases to consist almost entirely of metallic iron. Similar but much larger masses, which are presumably of similar origin, up to several tons in weight, have been found in the drier regions of the earth, such as Mexico, Peru, and Central Asia. These so-called *meteorites* are essentially iron free from carbon, often containing a notable proportion of nickel, and numerous compounds, such as ferrous sulphide (troilite), sulphide of chromium (daubréelite), calcium sulphide (oldhamite), and phosphide of iron and nickel (schreibersite), which are not known as terrestrial minerals; besides magnetic pyrites, chromic iron, magnetite, pyroxene, olivine, and anorthite, which are common constituents of volcanic rocks. In many meteorites the whole mass is made up of iron in a compact state, which, when polished or etched, develops the structure of interlacing plates known as Widmannstätten's figures, while in others it is associated with the siliceous minerals above named in various proportions, in more or less connected masses or diffused in grains. The latter condition is reproduced in certain basalts, such as those of Antrim, which, according to Andrews, contain grains of metallic iron, as do also some of the lavas of Auvergne. The surface of meteorites is often covered with a brilliant siliceous coating resembling a black varnish, caused by superficial fusion; the mass becoming intensely heated from the sudden retardation experienced on entering the earth's atmosphere. Among the more remarkable masses of meteoric iron may be mentioned that of 1,600 lbs. discovered by Pallas in Siberia; one in Brazil of 12,000 lbs.; one found in the Gran Chaco or interior plain of South America of 16,000 lbs.; and that of Durango in Mexico, weighing about 20 tons. Several remarkable masses have also been found at Ovifak in Greenland, the largest about 28 tons, but these are now supposed to be of terrestrial origin and to be derived from a mass of basalt.

Metallic iron is sometimes found when coal seams have taken fire in the vicinity of beds of iron ore. The product is melted into globules or buttons with a granular fracture resembling hard cast-steel. A mass of this kind was found some years since in Central France, where a coal seam was formerly on fire for some time.

The following analyses give the composition of meteorites from three different localities :—

ANALYSES OF METEORITES.

| | 1. | 2. | 3. |
|---|---|---|---|
| Fe | 88·042 | 90·153 | 83·02 |
| Ni | 10·732 | 6·553 | 14·42 |
| Co | 0·455 | 0·502 | 0·50 |
| Mn | 0·132 | 0·145 | ... |
| Cu<br>Sn | 0·066 | 0·080 | ·06 |
| Mg | 0·050 | 0·082 | MgO 0·24 |
| C | 0·043 | ... | ... |
| S | trace | 0·482 | 0·08 |
| F, N, P | ... | 1·226 | P 0·19 |
| Cl | ... | ... | 0·02 |
| Gangue | 0·480 | ... | $SiO_2$ 0·84 |
| | 100·000 | 99·223 | 99·37 |

1. From Siberia; by Berzelius. 2. From Lenarto; by W. S. Clarke. 3. From Knoxville, Tennessee; by J. L. Smith.

The ores of iron may be most conveniently considered at first according to their mineral classification, with their composition when in the pure state; the actual composition on the large scale being subsequently given when describing their associations and distribution.

MAGNETIC IRON ORE.—*Magnetite, Fer Oxydulé, Magneteisen.*—This crystallizes in the cubic system, usually in octahedra or rhombic dodecahedra, with an imperfect octahedral cleavage; but granular and compact forms are more general than crystals. Colour and streak iron-black; sp. gr. 5–5·2; hardness 5·5–6·5. Characteristically magnetic, being easily attracted by the magnet, and sometimes having polarity; also forms natural magnets capable of supporting considerable weight, but these are rare. Composition $Fe_2O_3 \cdot FeO$, or $Fe_3O_4$, corresponding to iron 72·48 and oxygen 27·52 per cent.

Magnetite is most abundant in the older crystalline and slaty rocks, such as granite, gneiss, mica-, hornblende-, and chlorite-schist, and in crystalline limestones. Very perfect crystals are found embedded in chlorite-schist; they are also common in the form of interspersed grains in basalts and other igneous rocks.

FRANKLINITE.—Cubic, crystallizing in forms exactly similar to those of magnetite, which it otherwise resembles, except that it is less decidedly magnetic; has a brownish streak; sp. gr. 5·1. In composition it is essentially a magnetite, containing zinc and manganese oxides partly replacing both those of iron. The average of several analyses by Rammelsberg gives: iron 45·16, manganese 9·38, zinc 20·30, oxygen 25·16 per cent. This mineral is restricted to a few localities near Franklin, New Jersey, where it occurs in large quantities, associated with oxide and silicate of zinc, in a large deposit in metamorphic limestone. The mixed minerals are first treated for zinc, and the residues as manganiferous iron ore.

HÆMATITE.—*Specular Iron Ore, Fer Oligiste, Eisenglanz.* Rhombohedral.—Crystals often combinations of rhombohedra and scalenohedra, or tabular hexagonal plates, of a dark steel-grey colour and metallic lustre. Opaque, except when very thin, when they transmit a brownish red light. Streak cherry-red. Sp. gr. 4·8 to 5·3. Hardness 5·5–6·5. Composition, $Fe_2O_3$, corresponding to iron 70 and oxygen 30 per cent. In addition to the crystallized forms, there are numerous other kinds, distinguished by special names. The most important of these is *red hæmatite* proper, *kidney ore*, or *Rother Glaskopf*, which occurs in uniform masses, showing a fibrous radiated structure when broken. The outside is very lustrous, but the fracture often dull. *Red ochre* and *puddler's ore* are soft earthy varieties of a bright red colour. *Micaceous iron ore* is a variety in thin crystalline scales, very slightly coherent, which soil the fingers like blacklead. *Oligiste, iron glance,* and *specular* or *looking-glass ore* are the hardest crystalline kinds; the latter is found in Brazil, in plates several inches across.

Hæmatite is found both in crystalline and stratified rocks, in the former often associated with magnetite. The best crystallized examples are from Elba, Brazil, and St. Gothard. Kidney ore is largely developed in irregular deposits in the Carboniferous Limestone. Very brilliant crystals of specular iron are also found in the lavas of modern volcanoes; the finest examples are those from the Island of Ascension. The softer varieties are sometimes ground, and used as paint for covering ironwork.

MARTITE is a mineral of a problematic character, having the octahedral form of magnetite, and the composition and physical characters of hæmatite. It occurs in considerable quantity in the Marquette district of Lake Superior, and also in Brazil and a few other localities. It is now generally considered to be a pseudomorphous variety of magnetite.

ILMENITE.—*Titaniferous Iron Ore, Fer Titané, Titaneisen.* Rhombohedral.—Crystals almost identical in form and dimensions with those of hæmatite, but with hemihedral development. Generally massive, of a dead black colour, and peculiar, even, non-granular fracture. Sp. gr. 4·5 to 5. Hardness 5–6. It contains ferric oxide and titanium dioxide in varying proportions, ferrous oxide, and invariably some magnesia. The composition varies considerably. Iron from 37 to 68, titanic oxide 17 to 58, magnesia 2 to 15 per cent. When the titanium is low, it passes into a titaniferous magnetite or hæmatite.

The characteristic localities of ilmenite are in the older rocks of Norway, Canada, and the Northern United States. It is also found interspersed in igneous rocks, which, by their waste, produce the heavy black sands common on many rivers and sea-beaches in different countries.

TURGITE. — *Hydro-hæmatite, Turgilite, Turgit.*—This, the lowest hydrate of ferric oxide, is of the composition $2Fe_2O_3.H_2O$, corresponding to ferric oxide 94·7 (iron 66·3), water 5·3 per cent. Specific gravity 3·74. Hardness 5.

It resembles massive red hæmatite, though somewhat greyer or browner in colour, having a fibrous radiated structure, and gives a bright red streak. Some varieties with the structure of hard kidney hæmatite decrepitate violently when heated. It is generally included with hæmatite, but is probably common as an independent mineral. Much of the soft red ore of Bilbao appears to be of this composition, and it is also found in Brazil, the Urals, and Connecticut.

GÖTHITE.—*Wood Iron Ore, Gœthite, Nadeleisenerz.*—Crystallizes in rhombic forms analogous to those of the similarly constituted minerals diaspore and manganite; they are very brilliant, in various shades of reddish-brown to nearly black. These crystals, by transmitted light, are brown to blood-red. It also occurs in mammillated, radial, fibrous forms, like those of red hæmatite, but brown or yellow in tint (wood iron ore). Streak brown or reddish brown. Specific gravity 3·8–4·4. Hardness 5·5. Composition $Fe_2O_3 \cdot H_2O$, corresponding to ferric oxide 90 (iron 63); water 10 per cent. The darker-coloured varieties often contain manganese.

Göthite is essentially a product of the alteration of other iron ores, particularly hæmatite and spathic ores. The purest crystallized examples have been obtained from Restormel in Cornwall, Lake Superior, and the Thuringian mines.

LIMONITE.—*Bog Iron Ore, Brown Hæmatite, Fer Oxydé Hydraté, Brauneisenstein, Raseneisenerz.*—The above are some of the names applied to the most abundant and least exactly defined among iron ores—the ferric hydrates with a maximum of water. In the richest form they approximate to the composition $2Fe_2O_3 \cdot 3H_2O$, corresponding to ferric oxide 85·6 (iron 60), and water 14·4 per cent., or the same as rust oxide. But generally they contain other bases, and sometimes silica, manganese oxides, phosphoric, sulphuric, and arsenic acids, and in bog ores organic matter; so that the proportion of iron is, as a rule, considerably lower. The colour is usually some kind of rusty brown, blacker when manganiferous, and the streak yellowish brown.

Being a product of the alteration of other minerals, it very commonly assumes their forms, notably those of cubes of iron pyrites in mineral veins, and of oolitic limestones and their fossils in the great Jurassic iron ore deposits of England, Luxembourg, and Lorraine. Apart from these, the most common mode of occurrence is in stalactitic or slag-like masses, and concretionary forms, often hollow. Bog iron ores have a peculiar resinous lustre. Pisolitic or bean ores are found in nodules, having a polished surface about the size of a haricot bean, filling pipes and funnels in limestone rocks. The lake ores of Sweden are granular limonites deposited under water, which contain infusorial remains in considerable quantity. The pigments known as ochres and umbers are limonites of an almost impalpable texture, containing clay; which, when calcined at graduated temperatures, produce a great variety of yellow and brown colours. In this country limonite is usually known as brown hæmatite, or, in special cases, as hydrate of iron. The more general name of brown iron ore would in most cases be preferable. German smelters call argillaceous or earthy hæmatite *Thoneisenstein*, which is thus different in meaning from its literal English equivalent, clay ironstone, a term restricted to the nodular carbonates of the Coal-measures.

SIDERITE.—*Spathic Iron Ore, Sphærosiderite, Fer Spathique, Spatheisenstein.* Rhombohedral. — Crystals principally simple rhombohedra, often with curved faces, also hexagonal flat prisms. In large deposits in crystalline aggregates, varying from largely plated to finely granular in texture; also in masses without apparent crystalline structure. Under particular conditions it may be nearly white, but discolours very rapidly on exposure; generally fawn colour, changing to brown or nearly black. Streak colourless. The largely crystalline varieties have a pearly lustre; the finer-grained and compact kinds are duller. Specific gravity 3·0 to 3·85. Hardness 3·5. Composition $FeCO_3$, corresponding to ferrous oxide 62 (iron 48·2), and carbon dioxide 38 per cent. This represents the theoretical composition, which is not realized on a large scale, part of the ferrous oxide being invariably replaced by some other base, such as manganese oxide, lime, or magnesia. The manganiferous varieties are the most valuable, but are liable to contain sulphides of other metals, such as copper and lead. The calcareous and magnesian varieties pass into the substances known as brown-spar and dolomite as the iron diminishes. The non-crystalline varieties are *clay ironstone* or *ball ironstone* (sphærosiderite), which resembles nodules of hardened clay or earthy limestone, and is found in the shales of the Coal-measures; *clay-band ironstone*, of similar composition, but in connected beds rather than nodules, and *blackband ironstone*, containing carbonaceous matter sufficient for calcination. The two latter varieties are found in the Scotch coal-fields. Spathic ores are exceedingly susceptible of change, and are often altered to considerable depths into brown ores (hydrated peroxides), which are of a dark colour when resulting from manganiferous ores, and stalactitic, cellular, or spongy; the change being attended with a large increase of volume. Such ores are among the most valuable from their easy reductibility.

In addition to the native minerals or ores proper, certain ferruginous substances, obtained as accessory or waste products in other metallurgical processes, are utilized by the iron-smelter. The most important of these are *purple ore* or *blue billy*, certain slags known as tap- and flue-cinders, and the scale or oxide produced in forges and rolling-mills. The first of these is the residue obtained from the treatment of cupriferous iron pyrites by the method of wet extraction, and consists essentially of ferric oxide in a nearly pure state, the principal impurity being a little sulphur, mainly as sulphate of lead. Considered as an iron ore, it is exceedingly rich, but the powdered state in which it is delivered to the smelter somewhat detracts from its value. It is chiefly used in the puddling process.

Tap- and flue-cinders are essentially ferrous silicates, the former produced in the puddling process, and the latter in re-heating and welding-furnaces. Roll- and hammer-scale are essentially magnetic oxides of iron; they are chiefly used in the forges, where they are produced in making up puddling-furnace bottoms.

## Distribution of Iron Ores.

The ores of iron, the composition of which has been indicated in the preceding paragraphs, are on the large scale associated with foreign minerals, which often materially influence their commercial value. Thus, a rich ore may be rendered almost valueless by the admixture of a notable quantity of calcium phosphate or of iron pyrites, while, on the other hand, a few units of manganese in a spathic ore, or of coaly matter in an argillaceous carbonate, though diminishing the absolute contents of iron, may add considerably to the selling price. It is therefore necessary to consider these associations somewhat in detail, by having in review the characteristics of the principal iron-producing districts. Ores of iron are found in rocks of every geological period, from the Archæan gneiss and granites, forming the base of the geological series in America, to the newest Post-tertiary, forming the great plains of both the northern and western continents; but the more important deposits may, speaking generally, be said to be confined to a few principal horizons, as follows :—

1. Archæan gneiss and schists. Most of the magnetites and schistose hæmatites of North America, Brazil, Scandinavia, India, and Eastern Algeria.
2. Silurian period. The oolitic hæmatite of the Clinton series in North America is the most considerable representative; the magnetite of Utah, and spathic ore of Styria and Carinthia.
3. Devonian period. Spathic ores of the Lahn and of North Devon. Red hæmatite of Nassau.
4. Carboniferous period. Red hæmatites of the Mountain Limestone in Cumberland and Lancashire. Clayband and blackband ironstones of Coal-measures in Great Britain and Westphalia.
5. Liassic and Oolitic periods. Oolitic carbonates of Cleveland.

Oolitic limonites of Northamptonshire, Luxembourg, &c. Red hæmatites of Southern France and Western Algeria.

6. Cretaceous period. Red and brown ores of Biscay.
7. Tertiary period. Aluminous limonites of Southern France and of Antrim.
8. Post-tertiary period. Lake and bog ores of Sweden, Finland, and North Germany. Pisolitic and granular ores of South Germany and Central France. Laterite of India and South America.

MAGNETIC ORES.—These ores, which, when massive, are often associated with hæmatite, are to a great extent confined to the older crystalline rocks of Scandinavia and North America, and generally occur in irregular beds in hornblendic and chloritic schists, in crystalline limestones, or as irregular veins and masses in rocks of eruptive origin. In the Laurentian rocks of Canada this ore is abundantly found in the gneiss and metamorphic limestones of the basin of the Ottawa; it usually occurs in the form of irregular beds, which, although not of any great lateral extent, are often of considerable thickness. Although of excellent quality, they have not been, as yet, extensively worked. One of the largest deposits of iron ore in Europe is probably that of Gellivara, in Swedish Lapland, situated about ninety miles from the head of the Gulf of Bothnia. According to the description of Erdmann and others, it forms a bold hill rising out of swampy ground, and consists of a number of parallel bands of magnetic and specular iron ores interlaminated with hornblendic and quartzose rocks. Some of these beds are between one hundred and two hundred feet in thickness, and may be traced for a distance of nearly seven hundred yards. Calcium phosphate is present in some of the beds, while others are apparently free from this impurity; iron pyrites appears to be almost entirely absent. The ore, on an average, contains about 90 per cent. of magnetic oxide of iron, and 3½ per cent. of silica. In some of the bands phosphoric acid is present to the extent of nearly 2 per cent.; others, on the contrary, contain only traces of this substance. These ores have been known for a very long period, but from the inaccessible nature of the country in which they occur, little use has been made of them; a railway to these mines from the Atlantic coast is now, however, being made.

The celebrated mines of Dannemora, in Sweden, are situated on the lake of the same name, about thirty miles from Upsala. The ore, which is a fine-grained magnetite, occurs in an irregular belt a mile and a half in length, and is specially employed for the manufacture of iron of the highest class. It occurs in crystalline limestone and felsite, and the workings extend to a depth from the surface of more than a hundred fathoms; the annual production does not, however, exceed 25,000 tons.

The following analyses show the composition of magnetic iron ores from various foreign localities:—

ANALYSES OF MAGNETIC IRON ORES—FOREIGN.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| $Fe_2O_3$ . . . . . . . | 58·93 | 62·06 | 56·80 | 81·16 |
| $FeO$ . . . . . . . | 27·55 | 28·42 | 27·50 | 11·06 |
| $MnO$ . . . . . . . | 0·10 | ... | 0·24 | 2·38 |
| $CaO$ . . . . . . . | 0·38 | trace | 1·80 | ... |
| $MgO$ . . . . . . . | 0·61 | 1·44 | 0·80 | ... |
| $Al_2O_3$ . . . . . . . | 0·29 | ... | ... | 0·60 |
| $S$ . . . . . . . | 0·04 | 0·07 | ... | ... |
| $CO_2$ . . . . . . . | 0·12 | ... | ... | ... |
| $H_2O$ . . . . . . . | 0·11 | ... | ... | 2·10 |
| $SiO_2$ . . . . . . . | 12·54 | 7·60 | 13·20 | 2·50 |
| $P_2O_5$ . . . . . . . | trace | ... | ... | ... |
| | 100·67 | 99·59 | 100·34 | 99·80 |
| Metallic iron . . . . . | 62·60 | 65·60 | 61·16 | 63·02 |

No. 1, from Dannemora, containing minute traces of iron pyrites; by Ward. Nos. 2 and 3, from Roslagen, on the east coast of Sweden, north of Stockholm; by Noad. No. 4, from Mokta-el-Haddid, Algeria, containing both magnetite and hæmatite.

Deposits of magnetite have been worked for several centuries in the neighbourhood of Arendal, in Norway, where they extend in a line nearly parallel with the coast for a distance of about thirteen miles. The ore occurs in hornblendic and micaceous schists, and chiefly consists of magnetite without any admixture of hæmatite.

On Lake Wetter, deposits of magnetite, in the form of strings and masses, are found in the Taberg, disseminated through porphyritic rocks. They form a hill of more than 350 feet in height, and, although comparatively of low produce, yielding only about 25 per cent. of metal, yet fuel being cheap in the district, they are worked with advantage. The metal produced is of good quality, and is especially adapted for wire-drawing. In the Ural, magnetite occurs under somewhat similar conditions to those which characterise the deposits in the vicinity of Lake Wetter. At Nischne-Tagilsk, a ridge of rock 300 fathoms in length, 250 broad, and about 40 in height, is, for the most part, made up of pure magnetite.

The Cerro Mercado, near Durango, Mexico, a hill 300 feet in height, is in great part composed of massive magnetite associated with specular iron ore, brown hæmatite, quartz, and calcite. Magnetic ores are found in large quantities in the Adirondack and other mountain regions of Northern New York and in the adjacent parts of New Jersey. Among the more important of these deposits are those of Crown Point and Lyons Mountain, near Lake Champlain. At the latter place a bed-like deposit about 16 feet thick has been worked on the dip for several hundred feet below the surface, and has been traced for about two miles along the course. The ore, a rich magnetite in felspathic gneiss, is not so free from phosphorus as that of Sweden, and is largely used in the direct production of malleable iron blooms in the open fire. The Cornwall ore-bank in Eastern Pennsylvania consists of alternations of magnetite with serpentine and chloritic schists. The ore averages 50 per cent., and is free from phosphorus. The out-put is the largest of any single mine in America. Large deposits of magnetite have been described by Newberry in Southern Utah, in shales supposed to be of Lower Silurian age Magnetite is one of the component minerals of the great schistose ore deposit of Mokta-el-Haddid, near Bona, in Algeria, which has for many years been one of the most productive mines of the world, but, as the analysis shows, the bulk of the ore is hæmatite. Magnetic ores of high quality occur near Salem in Madras, and were formerly smelted to some small extent for export. There are also important masses of magnetic and specular ores in the Chanda district in the Central Provinces of India, which both in composition and mode of occurrence are very similar to the Algerian ores.

In this country magnetic iron ores are of comparatively rare occurrence, but near Brent, in South Devon, magnetite is found in diorite, forming a deposit of about a foot in thickness; it also occurs on Dartmoor; at Treskerby, in Cornwall, it is found in a vein associated with tin ore.

The following analyses give the composition of three specimens of English magnetic iron ore:—

Analyses of Magnetic Iron Ores—British.

| | 1. | 2. | 3. |
|---|---|---|---|
| $Fe_2O_3$ . . . . . . | 62·20 | 44·40 | 66·50 |
| FeO . . . . . . | 16·20 | 20·00 | 13·00 |
| MnO . . . . . . | 0·14 | 0·16 | 0·56 |
| $Al_2O_3$ . . . . . . | 2·28 | 5·20 | 3·60 |
| CaO . . . . . . | 2·34 | 0·60 | 0·56 |
| MgO . . . . . . | 0·37 | 1·00 | 1·52 |
| $SiO_2$ . . . . . . | 0·24 | ... | ... |
| $P_2O_5$ . . . . . . | 0·10 | 0·50 | 0·57 |
| $FeS_2$ . . . . . . | 0·07 | $SO_3$0·04 | 0·04 |
| $H_2O$ hygroscopic . . . | 0·28 | 2·50 | 3·20 |
| $H_2O$ combined . . . | 0·34 | | |
| Insoluble residue . . . | 16·26 | 24·20 | 9·40 |
| | 100·82 | 98·60 | 98·95 |
| Metallic iron . . . . | 57·01 | 46·68 | 56·66 |

1. From Dartmoor, Devonshire; analysed by Riley. On operating on 900 grains of the mineral a solution was obtained in which minute traces of bismuth, tin, and copper were detected.[1] 2 and 3, from Cornwall; by Dr. Noad.

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[1] Percy's 'Metallurgy; Iron and Steel,' p. 224.

Many crystalline rocks contain magnetite and titaniferous iron ore interspersed in small grains and crystals. Such rocks when disintegrated give rise to the *black sands* which are abundantly found in alluvial gold-diggings and elsewhere. Along the sea-shore of countries in which crystalline and eruptive rocks abound, streaks of black sand, washed out by the action of the waves, are frequently found in sufficient quantity to admit of their being employed as an ore of iron. Among these localities may be mentioned the shores of the Bay of Naples, Taranaki, in New Zealand, and many points of the north-east coast of British America; the most important deposits are situated along the north shore of the St. Lawrence, east of the Moisie River. At Moisie, where these sands form a large portion of the beach, after the prevalence of certain winds, belts of nearly pure black sands, which have been concentrated by the action of the waves, are found along the shore. The purer and thicker layers are separated by shovels from the more siliceous portions, and are afterwards further concentrated by washing, whereby the siliceous matter is reduced to an average of about 5½ per cent.

When thus freed from the minerals with which they are associated these black sands consist of nearly equal portions of magnetite and titaniferous iron; those treated at the Moisie Iron-Works have been examined by Sterry Hunt, who found the magnetic portion to yield 66·73 per cent. of metallic iron and 4·15 per cent. of titanic oxide. The non-magnetic portion yielded 48·85 per cent. of iron and 28·95 per cent. of titanic oxide. A mixture of the two, as found, gave 55·23 per cent. of iron and 16 per cent. of titanic oxide. Similar iron sands occur at several points along the coast of Labrador, where they are associated with quartz, garnets, and felspar.

On account of their fine state of division, iron sands have not been successfully employed for the production of cast-iron, but at the Moisie works they were directly converted into wrought-iron in a bloomery-furnace, with satisfactory results.

Red Iron Ores.—These are frequently associated with the hydrated peroxide, particularly near the surface of deposits; where hæmatite is found in the newer rocks, the whole mass is often more or less mixed with brown iron ores. The most important deposits of these ores are found in Cambrian, Silurian, Devonian, and Carboniferous rocks. At Dalkarlsberg, near Nora, in Sweden, and in the island of Utö, specular hæmatite occurs with magnetite. In Saxony red iron ores are found near Eibenstock and Schwarzenberg, in lodes at the contact of mica-schist, altered Silurian rocks, and granite. Some of these lodes are as much as 15 fathoms in width, and extend for a distance of nearly 12 miles.

The iron mines of Elba, which are alike celebrated for the length of time they have been worked and for the beauty and purity of their ores, are, for the most part, situated near the eastern extremity of the island. At Rio Marina, specular and massive hæmatites rest upon talcose schists, and are covered by crystalline limestone. At Rio Albano and Terra

Nera the ore occurs in lodes, traversing talcose schists, which send off branches alternately coalescing into beds, some of which are 100 feet in thickness. A similar ramifying lode, which produces both hæmatite and magnetite, is seen in a limestone cliff at Cape Calamita. These ores are now considered to be thermal-spring deposits, and the quantity remaining is estimated at about 7,000,000 tons.

Near Marquette, on the southern shore of Lake Superior, a schistose variety of hæmatite is very extensively developed in Huronian rocks. The average breadth of this iron district is six miles, and it extends westwards from the lake shore for about twenty miles. The strata, which are much contorted, are chiefly talcose and chloritic schists, passing upwards into parallel laminæ of red jasper and hæmatite, whose united thickness is said to be upwards of 1,000 feet. Of this, a large proportion is too highly siliceous to pay for working; but distinct beds of 150 feet in thickness are quarried at the Jackson and Superior Mines. These deposits are, in addition to their purity and great extent, remarkable for containing minute crystals of martite. The same ores are found to extend into Wisconsin, the Menominee region on Lake Michigan being now nearly equal in importance to Marquette.

Two celebrated masses of hæmatite, known as the Iron Mountain and Pilot Knob, are worked near St. Louis, in Missouri. The Iron Mountain is a flattened dome-shaped elevation about 200 feet in height, and forms the western extremity of a ridge of reddish porphyry, which rises considerably above the iron ore, and stretches for more than a mile to the eastward. The surface of the mountain is entirely covered with loose pieces of ore, which become more and more conspicuous towards the summit. Moss-grown blocks, some of which are many tons in weight, cover the top, and are piled together in the greatest confusion. Pilot Knob is much higher than the Iron Mountain, its height above its base being estimated at 650 feet; it is mainly composed of a distinctly bedded siliceous rock. For the first two-thirds of the distance to the summit quartz rock predominates; the upper portion of the mountain consists of thick beds of iron ore alternating with siliceous rocks. The richest ores exhibit a distinctly slaty structure, differing entirely, in this respect, from those of the Iron Mountain, which are compact and without cleavage. There are numerous other localities in the vicinity, which, although not so generally known, yield ores of a quality at least equal to those obtained from Pilot Knob. The ore of Shepard's Mountain is magnetic and of high quality; the Russel-Bank ore is a fine-grained hæmatite, very pure, and making excellent iron.

Probably the largest masses of specular ore in the older rocks are those of Brazil, which, according to recent accounts, are developed on a scale unparalleled elsewhere, but at present they are almost unutilized. They are interesting as containing in places gold and auriferous pyrites in quantity.

The most important deposits of hæmatite in England are those of

Ulverstone in Lancashire and Whitehaven in Cumberland, which occur in the form of irregular masses in Carboniferous Limestone. The ore is usually in dull, compact masses, but also forms kidney-shaped crystalline aggregates. In addition to the compact variety there is a greasy micaceous ore, which is largely employed for lining the hearths of puddling-furnaces. Brown hæmatite appears to be entirely absent in both districts, and iron pyrites and calcium phosphate are only present in the most minute proportions.

ANALYSES OF RED IRON ORES.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| $Fe_2O_3$ . . . . . . | 95·16 | 90·36 | 86·50 | 94·23 |
| MnO . . . . . . | 0·24 | 0·10 | 0·21 | 0·23 |
| $Al_2O_3$ . . . . . . | ... | 0·37 | ... | 0·61 |
| CaO . . . . . . | 0·07 | 0·71 | 2·77 | 0·05 |
| MgO . . . . . . | ... | 0·06 | 1·46 | trace |
| $CO_2$ . . . . . . | ... | ... | 2·96 | ... |
| $P_2O_5$ . . . . . . | trace | trace | trace | trace |
| $SO_3$ . . . . . . | trace | trace | 0·11 | 0·09 |
| $FeS_2$ . . . . . . | trace | 0·06 | ... | 0·03 |
| $H_2O$ { hygroscopic . . . | ... | ... | ... | 0·39 |
| $H_2O$ { combined . . . . | ... | ... | ... | 0·17 |
| Insoluble residue . . . . | 5·68 | 8·54 | 6·55 | 5·18 |
| | 101·15 | 100·20 | 100·56 | 100·88 |
| Metallic iron . . . . | 66·60 | 63·25 | 60·55 | 65·98 |

1 and 2. From Cleator Moor, Whitehaven, Cumberland; by A. Dick. 1 is a compact red hæmatite containing cavities lined with crystals of specular iron and quartz. When large quantities were operated on, minute traces of lead were obtained. 2. A compact and pulverulent unctuous red hæmatite, in which traces of lead were detected. 3. Gillbrow, Ulverstone, Lancashire; by A. Dick. This ore is an unctuous red hæmatite, intermixed with pieces of limestone, which, being coloured red, cannot be seen until the specimen is washed. 4. Lindale Moor, Ulverstone; by J. Spiller. A hard compact hæmatite, affording distinct traces of arsenic when large quantities are operated on.

OLDER BROWN IRON ORES.—Irregular masses of brown hæmatite are met with in the Carboniferous Limestones and Lower Coal-measure sandstones in the Forest of Dean, in the neighbourhood of Bristol, and at Llantrissant in Glamorganshire. In the Forest of Dean the ore is a stalactiform brown hæmatite. The brown hæmatite of Ashton Court, near Bristol, frequently contains fragments of sulphate of barium porphyritically embedded. At Llantrissant the ore is interstratified between the upper part of the Carboniferous Limestone and a black shale roof, supposed to be a portion of the Coal-measures.

ANALYSES OF BROWN IRON ORES.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| $Fe_2O_3$ | 90·05 | 89·76 | 59·05 | 52·83 |
| $MnO$ | 0·08 | 0·04 | 0·09[1] | 0·81 |
| $Al_2O_3$ | trace | 0·63 | trace | ... |
| $CaO$ | 0·06 | 0·49 | 0·25 | 14·61 |
| $MgO$ | 0·20 | 0·40 | 0·28 | 5·70 |
| $CO_2$ | ... | ... | ... | 18·14 |
| $P_2O_5$ | 0·09 | 0·13 | 0·14 | 0·32 |
| $SO_3$ | trace | trace | ... | 0·28 |
| $FeS_2$ | ... | ... | 0·09 | ... |
| $SiO_2$ | ... | ... | 34·40 | ... |
| Organic matter | ... | ... | ... | 1·30 |
| $H_2O$ hygroscopic | ... | ... | 0·24 | ... |
| $H_2O$ combined | 9·22 | 7·05 | 6·14 | 4·75 |
| Insoluble residue | 1·07 | 2·57 | ... | 0·04 |
| | 100·77 | 101·07 | 100·68 | 98·78 |
| Metallic iron | 63·04 | 62·86 | 41·34 | 36·98 |

1. *Black Brush Ore*, Forest of Dean, Gloucestershire; contains minute traces of copper; by A. Dick. 2. *Smith Ore*, Forest of Dean; a comparatively pure brown hæmatite; by A. Dick. 3. From Llantrissant, Glamorganshire; by E. Riley. 4. From Froghall, near Cheadle, Staffordshire; compact, homogeneous, and brownish-red in colour; by A. Dick.

Enormous quantities of iron ore are now exported from Bilbao, in Spain, to England, and to such parts of France, Belgium, and Germany as are accessible by water-carriage. These are derived from deposits of Upper Cretaceous age, included between sandstones and limestones at the three principal localities of Somorrostro, Galdames, and Ollargan. The ores are of three kinds, of which the average composition is as follows:—

ANALYSES OF BILBAO ORES.

| | 1. | 2. | 3. |
|---|---|---|---|
| $SiO_2$ | 5·550 | 1·880 | 8·50 |
| $Fe_2O_3$ | 80·800 | 90·500 | 77·85 |
| $Al_2O_3$ | ... | ... | 1·50 |
| $MnO$ | 1·132 | 1·200 | ... |
| $CaO$ | 2·482 | 0·634 | 0·50 |
| $H_2O$, and volatile | 8·981 | 5·800 | 10·60 |
| | 98·945 | 100·014 | 98·95 |
| Metallic iron | 57·75 | 62·44 | 54·50 |

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[1] Estimated as $MnO_2$.

1. *Campanil*, a moderately hard red ore associated with calcite, apparently an altered spathic ore. It passes downwards into—2. *Vena dulce*, which is richer and softer; this is apparently turgite, though called red hæmatite. 3. *Rubio*, a concretionary limonite formed along the outcrop of a bed of sandstone containing nodules of siderite.

NEWER BROWN IRON ORES.—In the secondary formations, and more particularly in the Liassic and Oolitic series, stratified iron ores are largely developed both in England and in the western part of Continental Europe, and at the present time these formations collectively supply the largest amount of the ores smelted in Europe. Prominent among these are the Cleveland ores, in the marlstone or Middle Lias, which are more particularly considered at p. 151, and those of the Inferior Oolite, dogger, or Brown Jura formations, of which the chief representatives are the Northamptonshire ores in this country, and those of Luxembourg and Lorraine in Continental Europe. The Northampton ore, which is best developed about Wellingborough and Kettering, has the structure of a fossiliferous oolitic limestone, which is completely transformed into limonite at the surface, but when followed in depth is sometimes green, and contains ferrous carbonate and silicate. Such unweathered portions also contain notably more phosphorus than the surface ore. Through the same formations extends an ore largely worked in the counties of Rutland and Leicester, partly for smelting on the spot, but more particularly for export to Staffordshire, Yorkshire, and other older iron-making districts. In North Lincolnshire oolitic ores are extensively developed. Those dug at Frodingham, near Brigg, are more calcareous than those of Northamptonshire. The brown iron ores of Wiltshire are found in newer formations, namely, in the Coral Rag at Westbury, and in the Lower Greensand at Seend, near Devizes. The former locality is alone worked at present.

ANALYSES OF BROWN IRON ORES FROM THE OOLITE, &c.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| $Fe_2O_3$ | 50·31 | 55·42 | 53·43 | 46·59 |
| FeO | trace | ... | ... | ... |
| MnO | 0·51 | 0·44 | 1·60[1] | 1·75 |
| $Al_2O_3$ | 7·25 | 5·32 | 4·19 | 4·94 |
| CaO | 11·76 | 4·25 | 0·84 | 12·95 |
| MgO | 0·62 | ... | 0·72 | 2·84 |
| $SiO_2$ | 0·22 | ... | 24·81 | 13·59 |
| $CO_2$ | 7·98 | 5·69 | trace | 4·06 |
| $P_2O_5$ | 1·28 | 2·87 | 0·87 | 1·05 |
| $SO_3$ | ... | ... | ... | ... |
| $FeS_2$ | 0·17 | ... | trace | ... |
| $H_2O$ hygroscopic | ... | 12·21 | 13·61 | 12·17 |
| $H_2O$ combined | 11·00 | | | |
| Insoluble residue | 9·33 | 12·56 | ... | ... |
| | 100·43 | 98·76 | 100·07 | 99·94 |
| Metallic iron | 35·37 | 38·79 | 37·44 | 32·61 |

[1] Estimated as $MnO_2$.

1. From Wellingborough, Northamptonshire, in the Inferior Oolite; by E. Riley. 2. Green unweathered Northamptonshire ore; by L. Brown. 3. From Seend, Wiltshire, Lower Greensand; by E. Riley. 4. Frodingham, Lincolnshire, average of all the beds.

Red and brown hæmatites occur in the Liassic and Oolitic rocks of France under nearly similar conditions to those observed in this country. The most important deposits are those near La Voulte, in Ardèche, where large quantities of a compact earthy red hæmatite are interstratified in marls belonging either to the Lias or to the Oxford Clay.

In Würtemberg and Bavaria the lower members of the Oolitic group, on the north-west side of the Swabian Alps, contain ores of a similar character; their maximum development is in the neighbourhood of Aalen and Wasseralfingen, where the thickness of the beds exceeds 18 feet.

The oolitic ores of Luxembourg and Lorraine occur under very similar conditions to those of Northamptonshire, the most important centres of production being at Esch in Luxembourg, Longwy in France, and in the neighbourhood of Metz. The ore, which is generally known by the local name of *minette*, is partly calcareous and partly sandy, and contains from 30 to 35 per cent. of iron, with about 1 per cent. of phosphoric anhydride. It is largely exported to Belgium and Germany, in addition to supplying numerous local furnaces.

In addition to the stratified deposits before noticed, irregular masses of loose concretionary brown hæmatite, called *Bohnerz*, are found filling cavities and long winding fissures in the Oolites of South Germany. These pisolitic concretions vary from the size of a pea to that of a walnut; the larger being generally less perfectly spherical in form than those of smaller size. The cementing material is usually a ferruginous clay, which may be removed by washing, leaving an ore containing, on an average, about 35 per cent. of iron.

The Wealden rocks in the vicinity of Boulogne yield sandy brown ores occurring in superficial deposits; these chiefly supply the iron-furnaces at Marquise.

A large deposit of nodular brown iron ores, containing manganese and intimately mixed with phosphates, which are derived from the waste of older Oolitic and Neocomian strata, occurs in the Upper Cretaceous series near Ilsede, in Hanover, and is largely smelted to produce pig-iron rich in phosphorus for the Thomas and Gilchrist process.

TERTIARY AND POST-TERTIARY IRON ORES.—In England the ores of this class are of very little importance, but in many parts of Europe they are abundantly developed and extensively wrought. Large quantities occur in the great plain of North Germany, which extends from the borders of Holland to the head of the Baltic; they are also found abundantly in France, where they are worked to a considerable extent. The deposits of Berry, especially those in the valley of the Cher, are in the Tertiary series, and consist of pisolitic ores disseminated in beds in the argillaceous rocks of the district. Tertiary and alluvial ores are also extensively employed in the Ardennes and in the department of the Marne. Oolitic and pisolitic ores are obtained from the alluviums of the Nivernais, and hydrated oxides and hæmatites are found in the superficial clays of Charente, Dordogne, Lot-et-Garonne, Lot, and Tarn-et-Garonne. These are of very considerable importance, and are sufficiently pure to admit of being worked in the Catalan forge. The workings are, for the most part, open to the day, and frequently descend to a depth of sixty or seventy feet. The sandy deposits of the Landes also contain beds of hydrated oxides and of bog ores. Burat estimates that one-third of the iron annually produced in France is derived from alluvial ores.

In Sweden, Norway, and Finland, large quantities of a variety of limonite known as *lake ore* are obtained by dredging from the bottom of shallow lakes. This ore occurs in granular concretionary forms, varying in size from linseed to masses of several cubic inches. These ores are collected during the winter months only, for which purpose a hole of some three or four feet in diameter is made in the ice, through which is lowered a perforated iron shovel attached to a long wooden handle. Ores of this description are continually forming, and localities known to have been entirely exhausted a quarter of a century previously have, on being re-worked at the expiration of that period, been found to afford workable deposits of several inches in thickness. The formation of these ores is said to be mainly due to organic agency; the iron being chiefly derived from the oxidation of pyrites and from the decomposition of such minerals as hornblende, augite, &c., containing ferrous silicates.

ANALYSES OF BOG AND LAKE ORES.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| $Fe_2O_3$ | 62·59 | 70·46 | 66·33 | 67·59 |
| FeO | ... | ... | 3·60 | ... |
| $MnO_2$ | 8·52 | ... | 0·75 | 1·45 |
| $SiO_2$ | ... | 13·04 | 2·80 | 7·81 |
| $Al_2O_3$ | ... | 5·88 | ... | 4·18 |
| CaO | ... | ... | ... | 0·47 |
| MgO | ... | ... | ... | 0·23 |
| $P_2O_5$ | 1·50 | ... | 0·12 | 0·18 |
| Sand | 11·37 | ... | ... | ... |
| Water and organic matter | 16·02 | 11·12 | 26·40 | 17·81 |
| | 100·00 | 100·50 | 100·00 | 99·72 |
| Metallic iron | 43·82 | 49·32 | 49·24 | 47·32 |

1. Bog ore from the neighbourhood of Lingen, Hanover; by Senft. 2. Pisolitic ore from the district of Kandern; by Schenck. 3. Bog ore from State of New York; by Karsten. 4. Lake ore from Flaten, Wermland, Sweden; by Svanberg.

Spathic Iron Ore.—Siderite.—This ore does not so frequently occur in large masses as do the various forms of ferric oxide, but it is nevertheless found in very considerable quantities in various European localities. Large quantities of spathose ores are annually raised from the Devonian rocks in the district of Siegen; the most important mine is that of Stahlberg, near Müsen, where a nearly vertical lode has been worked since the commencement of the fourteenth century. Its form is somewhat that of a wedge; its greatest width being 65 feet, and its longitudinal extension 480 feet; it is enclosed in clay-slate, and has been worked to a depth of 130 fathoms by means of a series of levels driven into the hill. Spathic ores also abound in the crystalline metamorphic rocks and talcose-schists of the Eastern Alps, as well as in limestones of Silurian or Devonian age.

In Styria, the celebrated Erzberg, or Ore Mountain, rises to the height of nearly 2,500 feet, and, although apparently consisting of a solid mass of siderite, it is, in reality, only capped by an arch of that mineral, to a depth varying from 200 to 600 feet, including a few interstratified schistose partings. The best ore, which occurs in the lower beds, is hard and crystalline, and of a brownish-yellow colour. Iron and copper pyrites, quartz, carbonate of calcium, and, more rarely, cinnabar, are among the associated minerals. The yield in 1882 was about 512,000 tons, and at least 50,000,000 tons are said to have been laid open by the workings.

The less considerable, but otherwise similar, deposits of Carinthia are situated at Hüttenberg and Lölling, north-east of Klagenfurt, and include numerous lenticular beds, the most productive of which is about 200 feet in thickness.

In Thuringia a large irregular mass of spathic iron, in Permian rocks, has been worked for the last 700 years; the principal sources are those of Mommel and Stahlberg, near Schmalkalden, where the ores are much disturbed by intruded porphyritic dykes. This deposit has been followed to a depth of 300 feet, and its known length is about a mile.

In England there are deposits of carbonate of iron in Weardale, in Durham, where it occurs in veins in the Carboniferous Limestone, associated with ores of lead and zinc. Siderite is also found at Perran in Cornwall, Exmoor in Devon, and at Brendon Hill, Somerset. For some years these ores were worked to a considerable extent and exported to South Wales for the production of spiegeleisen. The outcrop of veins of this mineral is invariably found to be converted, to a considerable depth, into brown hæmatite by the action of atmospheric air and water.

Analyses of Spathic Iron Ores.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| $FeO$ | 49·77 | 43·84 | 47·16 | 55·64 |
| $Fe_2O_3$ | 0·81 | 0·81 | ... | ... |
| $MnO$ | 1·93 | 12·64 | 10·61 | 2·80 |
| $CaO$ | 3·96 | 0·28 | 0·50 | 0·92 |
| $MgO$ | 2·83 | 3·63 | 3·23 | 1·77 |
| $CO_2$ | 37·20 | 38·86 | 38·50 | 38·35 |
| $P_2O_5$ | trace | ... | ... | ... |
| $S$ | 0·04 | ... | ... | ... |
| $H_2O$ | 0·30 | 0·18 | ... | ... |
| Insoluble residue | 3·12 | 0·08 | ... | ... |
| | 99·96 | 100·32 | 100·00 | 99·48 |
| Metallic iron | 38·95 | 34·67 | 36·75 | 43·27 |

1. Weardale, Durham; containing minute traces of copper; Tookey. 2. Brendon Hill, Somersetshire; Spiller. 3. Stahlberg, Müsen; Schnabel. 4. Erzberg, Styria; Karsten.

Clay Ironstones.—These ores, which are characteristic of the Carboniferous series, although found to a smaller extent in the Wealden and other formations, occur in thin beds, or more generally in nodules, in the shales of the Coal-measures.

The nodules essentially consist of ferrous carbonate, but always contain a notable quantity of clay or sand, with variable proportions of the carbonates of calcium, magnesium, and manganese. When freshly broken, the fracture has a light-grey, yellow, or bluish tint, but eventually becomes brown on exposure, through superficial peroxidation of ferrous oxide. The nodules are frequently so numerous as to coalesce into beds, and sometimes contain fossils, such as fish and remains of plants. They are often fissured in such a way as to suggest the idea of its being the result of contraction by drying; and the fissures, having been subsequently filled with mineral matter, have the appearance of veins, which often contain quartz, iron and copper pyrites, galena, blende, and calcite. At Dowlais, near Merthyr Tydvil, in Glamorganshire, the clay ironstone, in addition to the minerals above enumerated, contains hattchetine, or mineral tallow, while beautiful thread-like crystals of millerite, or sulphide of nickel, occur in the partially filled cavities. Clay ironstone is abundant in the coal-fields of North and South Wales, Staffordshire, Shropshire, Yorkshire, Derbyshire, and Scotland; while those of Northumberland, Durham, and Lancashire are almost entirely without it. This ore is frequently worked in conjunction with the coal with which it is associated, and is extracted through the same pits.

The yield of clay ironstone per acre necessarily varies in accordance with the thickness of the bed and the regularity with which the nodules are disseminated. Thus the Parkgate Old Black Mine, which has a thickness of 11 inches, yields 1,500 tons per acre; while the Clay Wood Mine, of exactly one-half the thickness, produces in an equivalent area the same amount of ore. In the South Wales coal-field there are, at least, seven distinct districts containing seams of ironstone; the number of beds in each varies from six to twenty-two, and many of these individual deposits are themselves subdivided into several distinct courses. They sometimes consist of a single layer of spheroidal concretions or balls, of all sizes up to a ton or more in weight, but the beds are not usually very thick, and this is the greatest drawback to their value.

ANALYSES OF CLAY IRONSTONES.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| $FeO$ | 35·38 | 39·38 | 45·35 | 52·04 |
| $Fe_2O_3$ | 1·20 | 1·24 | ... | ... |
| $MnO$ | 0·94 | 0·95 | 0·56 | 0·92 |
| $Al_2O_3$ | 0·80 | 0·82 | 0·61 | 1·30 |
| $CaO$ | 2·78 | 2·26 | 2·60 | 0·53 |
| $MgO$ | 2·22 | 3·72 | 1·22 | 0·85 |
| $SiO_2$ | ... | ... | 0·67 | ... |
| $CO_2$ | 25·41 | 29·38 | 30·21 | 32·31 |
| $P_2O_5$ | 0·48 | 0·47 | 0·46 | 0·21 |
| $SO_3$ | trace | ... | trace | trace |
| $FeS_2$ | 0·18 | trace | 0·20 | 0·13 |
| $H_2O$ hygroscopic | 0·74 | 0·68 | 1·64 | 0·46 |
| $H_2O$ combined | 1·11 | 1·41 | | |
| Organic matter | 0·23 | 0·54 | 1·59 | 0·51 |
| Insoluble residue | 28·00 | 19·35 | 15·87 | 11·14 |
| | 99·47 | 100·20 | 100·98 | 100·40 |
| Metallic iron | 28·76 | 31·82 | 35·74 | 40·84 |

1. *White Bed Mine*, Brierley, Yorkshire; Spiller. 2. *Thorncliffe, White Mine*, Parkgate, Yorkshire; Spiller. 3. *Pins*, Dudley, Staffordshire; Dick. 4. *Gubbin and Balls*, Bunker's Hill Colliery, Staffordshire; Dick.

BLACKBAND IRONSTONE.—This ore was discovered in Lanarkshire by Mushet in the year 1801, but did not come into extensive use until about 1830. It is a clay ironstone, usually of a dark-brown or black colour, containing carbonaceous matter. The blackband iron ores of Scotland contain from 20 to 25 per cent. of coaly matter, and from 30 to 40 per cent. of iron. In the western coal-field of Scotland several principal blackband measures, having an aggregate thickness of 6 ft. 5 in., are known; all these have been more or less extensively worked, but the supply has now considerably fallen off. The thickness of deposits of blackband is subject to great variation, and a band seldom extends over very large areas without some change taking place in its composition. At Airdrie the blackband occurs in workable quantity over an area of only about ten square miles; whilst its equivalent, in the form of a thin seam of coal, is found over a district of more than five times that extent. In Linlithgowshire the stratum, elsewhere affording blackband ironstone, is represented by the celebrated Boghead cannel coal.

The yield of blackband ironstone is at the rate of 2,000 tons of calcined ore per acre for each foot in thickness of the deposit. Beds varying from 4 to 9 feet in thickness occur in North Staffordshire, and, after calcination, the ore is largely exported to the iron-works of South Staffordshire.

In South Wales blackband is found in numerous beds of irregular thickness and of limited extent. This ore was first discovered in Westphalia, in 1855; but the quantity does not appear to be large, and in but a few instances only do blackband and coal occur together. Blackband likewise occurs in Lower Silesia, and thin layers of this mineral have been discovered at one or two localities in the Banat.

Carbonaceous spathic ores, locally known as *coal brasses*,[1] occasionally accompany coal in South Wales, and differ from ordinary blackband in containing considerable quantities of the carbonates of calcium and magnesium.

ANALYSES OF BLACKBAND IRONSTONES.

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| FeO | 53·82 | 40·77 | 37·07 | 42·64 |
| $Fe_2O_3$ | 0·23 | 2·72 | ... | ... |
| MnO | ... | ... | 0·23 | 0·26 |
| CaO | 1·51 | 0·90 | 6·61 | 5·24 |
| MgO | 0·28 | 0·72 | 7·40 | 5·26 |
| $SiO_2$ | 2·00 | clay 10·10 | 2·70 | ... |
| $CO_2$ | 34·39 | 26·41 | 36·14 | 36·89 |
| $P_2O_5$ | ... | ... | 0·23 | 0·17 |
| $FeS_2$ | ... | ... | trace | 0·22 |
| $H_2O$ hygroscopic | ... | ... | ... | ... |
| $H_2O$ combined | ... | 1·00 | ... | ... |
| Carbonaceous matter | 7·70 | 17·38 | 9·80 | 8·87 |
| | 99·93 | 100·00 | 100·18 | 99·55 |
| Metallic iron | 41·60 | 33·57 | 28·83 | 33·12 |

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[1] This term is generally applied in English collieries to nodules of iron pyrites found in the coals.

1 and 2. Scotch blackband; Colquhoun. 3 and 4. "Coal brasses," Aberdare, South Wales; Price and Nicholson.

**Cleveland Ironstone.**—The Middle Lias, or marlstone rock, which, in the neighbourhood of Chipping Norton and Woodstock in Oxfordshire, yields an oolitic brown hæmatite, affords iron ore in much larger quantities in the North Riding of Yorkshire. This stone, where best developed in the Cleveland district, has a total thickness of about 20 feet, made up of various interstratified bands of ore and shale; of these the two principal members are distinguished as the *Pecten* and *Avicula* seams, from the respective prevalence in them of fossil shells belonging to these genera. The average workable thickness of the ore is from 12 to 17 feet; the usual produce per acre being estimated at about 20,000 tons. This ore, which is an inferior carbonate of iron, is usually of a dull bluish-yellow colour, caused by the presence of ferrous silicate; its structure is oolitic, with numerous enclosed fossils, and it sometimes contains interspersed crystals of quartz and anatase. A dark blue or nearly black variety, found at Rosedale Abbey, is, although oolitic in its structure, both magnetic and polar.

**Analyses of Cleveland Iron Ores.**

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| FeO | 39·92 | 34·98 | 33·17 | 33·85 |
| $Fe_2O_3$ | 3·60 | ... | ... | 32·67 |
| MnO | ·95 | 0·48 | 0·50 | 0·69 |
| $Al_2O_3$ | 7·86 | 3·20 | 3·92 | 3·15 |
| CaO | 7·44 | 11·96 | 11·90 | 2·86 |
| MgO | 3·82 | 4·51 | 4·52 | 1·59 |
| $K_2O$ | 0·27 | ... | ... | ... |
| $SiO_2$ | 7·12 | ... | ... | 6·95 |
| $CO_2$ | 22·85 | 29·20 | 28·00 | 10·36 |
| $P_2O_5$ | 1·86 | 1·30 | 0·48 | 1·41 |
| $SO_3$ | trace | ... | ... | ... |
| $FeS_2$ | 0·11 | ... | ... | 0·03[1] |
| $H_2O$ hygroscopic | ... | 3·30 | 3·65 | 3·76 |
| $H_2O$ combined | 2·97 | | | |
| Organic matter | trace | ... | ... | 0·84 |
| Insoluble residue | 1·64 | 10·04 | 13·22 | ... |
| | 100·41 | 98·97 | 99·36 | 98·16 |
| Metallic iron | 33·62 | 27·21 | 25·80 | 49·17 |

1. Cleveland ore, locality not stated; Dick. 2. *Pecten Bed*, Grosmont, Yorkshire; Tookey. 3. *Avicula Bed*, Grosmont, Yorkshire; Tookey. 4. Magnetic ore, Rosedale Abbey; Pattinson.

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[1] Sulphur.

MANGANIFEROUS IRON ORES.—Since the introduction of the Bessemer and Siemens processes for steel-making, the production of manganese and of manganiferous pig-iron, which was formerly restricted to localities producing spathic ores, has become general in most of the larger iron-making districts in England, Belgium, France, and Germany, the necessary ores being often brought from considerable distances. These are chiefly limonites, more or less intimately mixed with peroxide of manganese ($MnO^2$) as pyrolusite, or in the less pure form of psilomelane, wad, &c., which contain in addition sulphate of barium, alkalies, &c. These occur in limestone districts producing lead ores, and probably represent the superficial alteration of manganiferous spathic ores. The more important localities are Almeria and Porman, near Cartagena, whence there is a large export, Laurium in Greece, South-Western Portugal, Nassau, and Carniola. The composition varies very considerably, the points of chief importance being a large amount of manganese as compared with iron, 25 to 35 per cent. of each, and a low proportion of silica, phosphorus, and sulphur; lime is not objectionable. Speaking generally, the qualities that render a manganese ore unfitted for the chemical manufacturer make it of value for spiegel-making. Pure pyrolusite, though it may be used, is, apart from its high price, not advantageous as a source of manganese, as it gives off oxygen at a low heat, and thus wastes fuel in the upper part of the furnace.

ALUMINOUS IRON ORES.—In some few cases limonites have been found which contain alumina uncombined with silica, and may therefore be used in the smelting of siliceous hæmatites when it is desired to employ alumina as a fluxing material. In this respect they stand midway between ores proper and fluxes. The most important localities are Les Beaux, near Toulouse, whence the mineralogical name Bauxite has been derived; Wochein in Carniola, and the neighbourhood of Larne in Antrim. In the latter district it occurs with pisolitic ores below the Antrim basalt in the beds of old lakes, and is largely exported as Belfast aluminous ores. The purer varieties are also used as a source of aluminium.

ANALYSES.

| | $SiO_2$. | $Al_2O_3$. | $Fe_2O_3$. | $FeO$. | $CaO$. | $MgO$. | $SiO_2$. | $H_2O$. |
|---|---|---|---|---|---|---|---|---|
| Belfast ore; Tookey | 9·87 | 34·57 | 27·93 | 5·08 | 0·91 | 0·62 | 3·51 | 19·36 |
| Bauxite; Bell . . | 2·80 | 57·40 | 25·50 | ... | 0·62 | ... | 3·10 | 11·00 |

The following is the return of the amount of iron ore raised and consumed in the United Kingdom for the year 1885:[1]—

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[1] 'Mining and Mineral Statistics of the United Kingdom of Great Britain and Ireland for the year 1885.'

TOTAL QUANTITIES OF IRON ORE AND IRONSTONE PRODUCED IN AND IMPORTED INTO THE UNITED KINGDOM IN 1885.

| District. | Description of Ore. | Tons. |
|---|---|---|
| ENGLAND. | | |
| Devonshire . . . | Brown Hæmatite and Magnetite . | 1,928 |
| Somersetshire . . | Spathose ore, Argillaceous ore, and Hæmatite . . . . . . | 2,099 |
| Gloucestershire . . | Oolitic ore . . . . . . | 45,125 |
| Wiltshire . . . | „ . . . . . . . | 67,489 |
| Rutlandshire and Oxfordshire . . . | „ . . . . . . . | 25,376 |
| Northamptonshire . | „ . . . . . . . | 1,160,000 |
| Leicestershire . . | „ . . . . . . . | 310,529 |
| Lincolnshire . . . | „ . . . . . . . | 1,188,524 |
| Shropshire . . . | Argillaceous Carbonate . . . | 177,620 |
| Warwickshire . . | „ „ . . . . . | 10,981 |
| Staffordshire, North . | Argillaceous Carbonate and Hydrated Oxide . . . . . . . | 1,743,300 |
| Staffordshire, South, and Worcestershire . . | Argillaceous Carbonate . . . | 117,726 |
| Derbyshire . . . | „ „ . . . . . | 18,405 |
| Lancashire . . . | Red Hæmatite and Argillaceous Carbonate . . . . . . . | 1,209,971 |
| Cumberland . . . | „ „ . . . . . | 1,228,323 |
| Yorkshire North Riding | Argillaceous Carbonate, and Magnetite . . . . . . . | 5,932,244 |
| Yorkshire West Riding | Argillaceous Carbonate . . . | 126,596 |
| Durham . . . | Spathose ore . . . . . . | 39,777 |
| NORTH WALES . . | Argillaceous Carbonate and Brown Hæmatite . . . . . . | 2,606 |
| SOUTH WALES AND MONMOUTHSHIRE . | Argillaceous Carbonate, Blackband, and Brown Hæmatite . . . . | 63,294 |
| ISLE OF MAN . . | Spathose ore . . . . . . | ... |
| SCOTLAND . . . | Argillaceous Carbonate, Blackband, and Red Hæmatite . . . . | 1,838,423 |
| IRELAND . . . | Aluminous ore, Blackband, and Brown Hæmatite . . . . | 107,646 |
| PURPLE ORE FROM CUPRIFEROUS PYRITES . | ... ... ... ... | 490,890 |
| IRON ORE IMPORTED . | ... ... ... ... | 2,822,598 |
| | Total . . . . . . | 18,731,470 |

## ASSAY OF IRON ORES.

In order to determine the commercial value of an ore of iron, it is necessary to ascertain not only its percentage yield of metal, but also the approximate proportion and constitution of the various impurities present, as well as their probable influence as affecting its fusibility. It is, moreover, requisite to determine the nature and respective amounts of other elements, such as phosphorus, sulphur, &c., likely to affect the quality of the iron obtained by its metallurgical treatment. The two former questions may be answered by the results of dry assay, which gives the maximum amount of cast-iron that, under the most favourable conditions, can be obtained from the ore in the blast-furnace; but for the last, the methods and processes of chemical analysis are required.

We shall first give a description of the methods of assaying iron ores by the dry way, and afterwards treat of the processes for the volumetric estimation of iron; finally, the analysis of ironstones and the estimation of those constituents which are present in minute quantities only, will be considered.

Dry Assay of Iron Ores.—This operation is conducted in a wind-furnace, such as is represented in fig. 28. This is a vertical section through the middle of the grate of one of the assay-furnaces employed in the metallurgical laboratory at the Royal School of Mines. The "melting-hole," A, 8 inches square and 12 inches deep, is lined with refractory bricks. Below the grate, *g*, is the ash-pit, B, closed in front by the door, *c*, which has a revolving register-plate for regulating the air-supply. The draught is maintained by a stack 60 feet high, connected with the furnace by the flue, D. The same stack serves five similar furnaces.

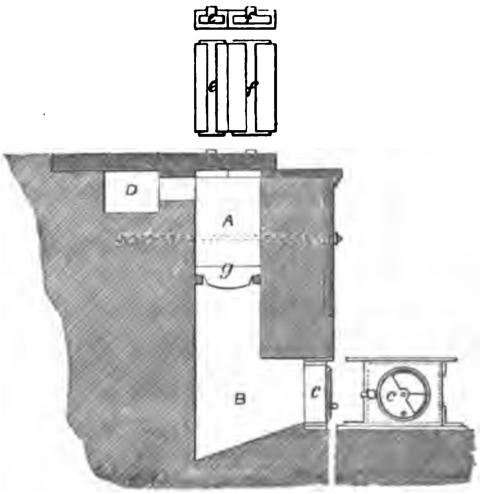

Fig. 28.—Assay-Furnace; vertical section.

The furnace is closed by two fire-tiles, *e*, *f*, clamped with iron, and wedged at one end. These are of unequal size; the larger one only is removed when a crucible is to be taken out of the furnace. A more usual covering arrangement is to have a square tile or "quarry" set in an iron frame, with a projecting handle. Besides the ash-pit door register, the draught may be regulated by a damper at the top of the stack, by a slide-valve, or by a loosely fitting piece of brick in the passage connected with the flue.

The brickwork, which is in all parts subject to high temperature, is made of fire-brick set in fire-clay, is faced with cast-iron plates and bound together by wrought-iron tie-rods. The fuel used may be either clean coke or anthracite. Charcoal is also convenient, but is now rarely attainable.

*Tongs.*—In conducting a dry assay an assortment of iron tongs of different shapes are required, some of which are represented in figs. 29–32. A short bent pair with scissor handles (fig. 29) is useful in filling in lumps of fuel when the fire has burnt hollow, and for handling crucibles after they have been taken out of the furnace. For removing these, tongs with straight handles, such as figs. 30, 31, are preferable, as giving a firmer grip. Fig. 30 has curved jaws for clasping round a crucible when it is covered and cannot be gripped at the top. All these, being rather short in the handles, are inconvenient for manipulating highly heated objects, and can only be used in the removal of crucibles when they have cooled down in the furnace, which may require about half an hour after the fusion is complete. This loss of time may often be prevented by having a longer handled pair of tongs, such as fig. 32, which can be used as soon as the crucible is sufficiently hardened to bear handling. To protect the face from the glare, and to better distinguish the position of the crucible in the heated furnace, a wooden shield, fig. 33, with a peep-hole, *a*, glazed with blue or green glass, and handle, *b*, may be used. This is held by an assistant before the face of the operator who is engaged in searching for the crucible.

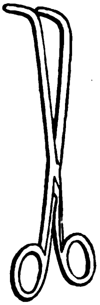
Fig. 29.

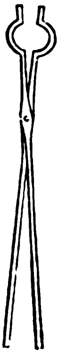
Fig. 30.

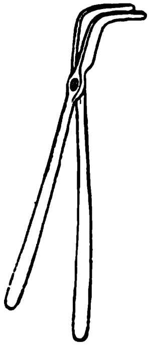
Fig. 31.

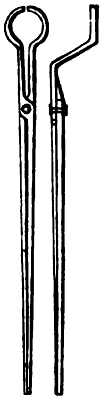
Fig. 32.

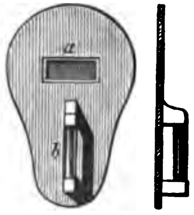

Fig. 33.

*Sampling.*—In all assaying operations the preparation of a sample which shall fairly represent the bulk of the ore under investigation is of first importance, and in order to do this a systematic plan must be followed. Supposing a quantity of about 14 lbs. weight, in lumps of the size delivered to the smelter, to be available, the whole quantity must be broken down to about ¼ or ½ inch fragments. This can readily be done upon an iron or steel plate with a flat-faced hammer, the lumps being placed in an iron ring somewhat larger than the hammer face to prevent the fragments flying apart. The broken stuff and dust are then carefully mixed into a circular heap, from which a portion is removed by cutting it completely through with a broad-bladed spatula in two directions at right angles to each other; this is known as the method of quartering. This portion, from ½ to 1 lb. in weight, is then further reduced, either by crushing or by pulverizing in a cast-iron mortar, until it passes through a sieve of 8 to 10 holes to the linear inch, and from this a final sample of 500 to 1,000 grains or more is taken. This should be reduced sufficiently to pass through a gauge of 60 to 80 holes to the linear inch. In powdering hard and quartzose ores in cast-iron mortars, a beating or percussive action of the pestle is necessary, and all triturating or grinding should be avoided, as by that means particles of metal are worn off and tend to increase the apparent amount of iron present. Limonite and other soft ores may be conveniently pulverized in Wedgwood-ware mortars, but for finishing the final sample an agate mortar is generally preferable. The finely powdered sample should be preserved in a stoppered bottle to prevent loss or undue increase of hygroscopic water.

The object of the dry assay is to obtain the iron of the ore as a compact button of cast-iron, the ore being reduced by carbon, and the earthy components fused down by the agency of appropriate fluxes, which will vary with the nature of the ore under examination. It is therefore necessary to have some preliminary knowledge of the composition of the mineral, and where this cannot be obtained by inspection or repute, a complete or partial chemical analysis is required. A method of the latter kind devised by Berthier is sometimes used. It includes the following operations:—

1. *Determination of Volatile Matter.*—About 50 grains of mineral are gradually heated up to bright redness over a Bunsen gas flame in a platinum or porcelain[1] crucible. The loss of weight gives the amount of water and carbon dioxide present, except where the ore contains ferrous compounds, when there is a certain increase of weight from the absorption of oxygen consequent on the formation of ferric oxide.

2. *Determination of Earthy Carbonates.*—Another portion of 50 grains is digested in the cold, either with acetic or very dilute nitric acid. This removes calcium and magnesium carbonates, while ferrous carbonates, if present, are not attacked. The operation is continued until

---

[1] Preferably the latter. Platinum vessels must be very cautiously used in iron analysis, for if the ore contains organic matter the reducing gases evolved may cause partial reduction of ferric oxide, and produce a superficial black stain of alloyed iron on the platinum, which can scarcely be removed by any reagent.

effervescence ceases, when the residue is washed, dried, and weighed. The amount of carbonate is determined from the loss.

*Determination of Insoluble Matter.*—50 grains of the ore are digested with strong hydrochloric acid, and boiled until the residue is colourless. This, when filtered off and washed, is calcined and weighed; it consists of quartz and silicates not decomposable by acids, and for every purpose may be regarded as silica.

The above method, though sometimes useful, is too inexact in its results to be of much value, having regard to the time required for carrying it out, and in most cases the nature of the fluxes required may be determined by inspection. Thus, the older magnetites and red ores are generally quartzose, and require lime and aluminous fluxes, while spathic and nodular carbonates, containing clay and lime, require a larger proportion of silica.

*Fluxes.*—The following are the principal fluxes required:—*Silica*, in the form either of white glass-house sand or ground flints: any other form of silica may be employed if practically free from iron, and in a sufficiently fine state of division. *Alumina;* this is best supplied in the form of china-clay, which, after ignition, contains 53 per cent. of silica, and 43 per cent. of alumina. Fire-clay, shale, or blast-furnace cinder may also be used, but they have the disadvantage of containing small quantities of iron. *Lime;* unslaked lime in a state of fine powder should be used, when obtainable, but chalk or any other limestone sufficiently free from iron may be employed as a substitute; care must be taken that this flux is as free as possible from sulphates and phosphates. In some cases *fluor-spar* may be advantageously substituted for lime. This flux affords a liquid slag, but is rarely used on the large scale. *Glass,* free from lead, may be employed with argillaceous ores; such glass usually containing from 60 to 70 per cent. of silica, the remainder being soda, potash, and lime. *Borax* is too fusible to admit of being generally employed as a flux for iron assays, since it combines with iron oxide at a temperature below that necessary for reduction. It is nevertheless sometimes used as a substitute for glass, but, when it is so employed, the proportion of lime should be increased to such an extent as to materially diminish the fusibility of the mixture.

The nature and quantity of flux required in an iron assay will vary with the proportion and character of the earthy matters in the ore. The object sought is to obtain a sufficient amount of well-fused, clean slag to completely cover the reduced button of metal. It must also be sufficiently refractory to allow the iron to become fully carburized before fusion.

According to Percy, grey blast-furnace cinder may be taken as a type of the kind of slag most desirable to obtain; its approximate percentage composition is as follows:—

| | | | |
|---|---|---|---|
| $SiO_2$ | 38 | or about | 2½ parts |
| $Al_2O_3$ | 15 | | 1 " |
| $CaO$ | 47 | | 3 " |

The following mixtures produce, when fused, slags approximating to this composition:—

| | | | | |
|---|---|---|---|---|
| Quartz | 1 | | | |
| China-clay | 2 | $SiO_2$ 0·92 | 1·92 | 36·5 per cent. |
| | | $Al_2O_3$ 0·82 | 0·82 | 15·5 ,, |
| Lime | 2½ | | 2·50 | 48·0 ,, |
| Glass | 2½ | $SiO_2$ | 1·75 | 35·0 ,, |
| | | Bases = $Al_2O_3$ | 0·75 | 15·0 ,, |
| Lime | 2½ | | 2·50 | 50·0 ,, |
| Shale, or fire-clay | 3 | $SiO_2$ | 1·30 | 35·0 ,, |
| | | $Al_2O_3$ | 0·90 | 17·0 ,, |
| Lime | 2½ | | 2·50 | 48·0 ,, |

In some German iron-works the following proportions are employed:—

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| For rich magnetic ores and red hæmatites | 5 to 20 | p. c. | chalk | and | 25 | p. c. | fluor-spar. |
| ,, argillaceous brown ores | 20 to 40 | ,, | ,, | ,, | 30 to 40 | ,, | ,, |
| ,, bog ores | 50 | ,, | ,, | ,, | 50 | ,, | ,, |
| ,, spathose ores | 10 to 15 | ,, | china-clay | ,, | 20 to 25 | ,, | ,, |
| ,, tap- and mill-cinder | 20 to 25 | ,, | chalk | ,, | 20 to 25 | ,, | ,, |

When the composition of the ore is unknown, the proper flux may be determined empirically by trial and error, three or four samples being heated in the furnace together with some of the mixtures in the following table:—

| | 1. | 2. | 3. | 4. | 5. |
|---|---|---|---|---|---|
| $SiO_2$ | 50 | 50 | 30 | 45 | 15 |
| $Al_2O_3$ | 25 | 16 | 20 | 18 | 5 |
| CaO | 25 | 34 | 50 | 37 | 80 |

The weight of flux used in each case must be one-half that of the ore operated on, and the most advantageous mixture will be that yielding the highest produce.

*Preparation of the Assay.*—The fusion may be effected either in a plain or in a carbon-lined or *brasqued* crucible. In the former case, from 100 to 150 grains of the ore sample are intimately mixed with the required amount of fluxes and about 25 per cent. of powdered charcoal, and placed in a black-lead or clay crucible, which when filled is covered with a clay lid luted round the edges with fire-clay. In the second case, which is generally preferable, a clay (London or Hessian) crucible is prepared by filling it with charcoal powder rendered plastic with treacle or sugar syrup, which is rammed in with a wooden pestle in successive layers. Care must be taken to roughen the surface of each layer, so as to allow the next to adhere to it properly. When carefully done, the whole will form a coherent mass. A cavity is then bored so as to leave a thickness of about half an inch of carbon around and below it, as in the section fig. 34. The sides and bottom of the cavity are carefully polished with a glass rod. When quite dry the cementing material is carbonized by covering the crucible and exposing it to a dull red heat, which may be conveniently done in the flue of the assay furnace or in a muffle. These linings, when properly made and fired, give a compact mass of charcoal of considerable strength. The assay sample and fluxes, mixed as in the first case, are charged into the crucible, filling the space *b*, *c*, but no charcoal is required, the reduction being effected by the lining. The top, *a*, *b*, of the cavity is filled either with charcoal powder, or preferably by a solid sponge of charcoal, and a cover is fitted on later.

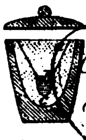

Fig. 34.

*Fusion of the Assay.*—The charged crucible when dried is placed in the centre of the furnace about 3 or 4 inches above the grate-bars, upon a stand made of a piece of fire-brick. One, two, or four assays may be made at once, according to the size of the crucibles employed; two is perhaps the most convenient number. After filling the furnace with fuel, the firing is conducted with the register nearly closed, in order to allow the crucibles to become gradually heated, and the water and volatile constituents to escape without cracking the luting. After about half an hour the draught is gradually increased, until at the end of an hour the furnace is urged to its highest temperature, which is maintained for a quarter or half an hour longer, after which time it is allowed to cool down. The crucibles, with the stands, to which they are generally fluxed by the slag formed by the ash of the fuel, are then removed and allowed to stand on the top of the furnace until cooled. The lid is then removed, and the crucible cracked in two lengthwise by a small chisel-faced hammer. If the operation has been successful, the iron will be found in a rounded button covered by a mass of well-melted glassy slag, which can be easily separated. Usually, however, in addition to the main button, some of the reduced iron will be found in shots through the slag, so that it is always necessary to break the latter to a coarse powder in a steel mortar, and separate the interspersed metal by passing a magnet through the powder. The particles so collected are added to the main button and weighed with it. If, from improper fluxing or insufficient heat, the ore has not been perfectly reduced, the metal will be found in the condition of malleable and not of soft cast-iron. A glassy transparent slag of dark green colour indicates that the earthy bases added have been insufficient to flux the silica, and that some iron has been scorified. On the other hand, lime and magnesia in excess give stony or crystalline masses, which are only fused, but not melted, having the iron diffused through them in crystalline scales. A light-grey or bluish enamel, or a white or smoky-grey translucent glass, is obtained when the fluxes have been properly proportioned. This will be coloured purple by manganese when in small proportion, but green or brown when in larger quantity. The button of iron when weighed should be broken between paper by striking it a smart blow upon an anvil. Under the most favourable conditions it will show slight malleability, and break with a dark-grey fine-grained fracture, like that of foundry pig-iron, which is rendered somewhat brighter and harder by phosphorus, and is spotted with white if sulphur is present. If any notable amount of manganese has been reduced from the ore, a brittle white metal of a platy crystalline structure is produced. If the metal is grey and well melted, the phosphorus in the ore will be completely reduced, and the button may be used for estimating phosphorus. The weight of the metal obtained by a dry assay should be somewhat more than that of the iron contained in the ore, as the return is made in cast-iron which contains several units per cent. of substances other than metallic iron. Practically, however, this is not always realized, there being generally some iron taken up by the slag, and the pig-iron yield will very nearly represent the amount of pure iron in the ore. If, however, the ore contains much manganese, the excess of weight may be from 4 to 7 per cent., or more.

Swedish Process.—In Sweden assays of iron ores are made in small brasqued crucibles, each about 2 inches in height, and 1¼ inch in diameter at its largest end. The weight of ore operated on is usually from 15 to 20 grains only, and four crucibles are placed in the furnace at the same time; a piece of fire-brick, about 3 inches square, being used as a stand. As soon as a white heat has been reached, the fire is allowed to burn down, and, when the furnace has sufficiently cooled, the crucibles are removed by lifting out the stands, to which they will be found firmly attached by the partial fusion of the brick with the ashes of the fuel employed. When cold they are broken, and the buttons and slags examined in the usual way; the four results, when the assays have been skilfully made, should not vary more than $\frac{3}{10}$ or $\frac{4}{10}$ per cent. from each other. As in the case of assays made upon larger quantities of ore, the metallic shot, when wrapped in stout paper and struck by a hammer on an anvil, should flatten slightly before breaking.

Wet Assay of Iron Ores.—Iron may be directly determined by bringing it into solution as a ferric salt, and precipitating it as ferric hydrate by means of potash or ammonia. The precipitate when washed, dried, and calcined at a red heat gives ferric oxide, containing 70 per cent. by weight of iron. This method, is, however inconvenient, not only on account of the gelatinous character of the precipitate, which renders the washing tedious and troublesome, but more particularly from the circumstance that ferric hydrate carries down phosphoric acid and silica with it; when these are present, even in small quantity, it cannot be separated from them by washing alone. The method of direct determination is therefore almost entirely abandoned in favour of more rapid indirect methods, which are susceptible of extreme accuracy. These depend upon the circumstance that when iron in solution as ferrous chloride is converted into ferric chloride by a cold solution of an oxidizing agent, the consumption of the latter is proportional to the amount of metal present, and if the solutions are of known strength, the amount can be determined by measuring instead of weighing. Two principal methods are in use, which are due to Marguérite and Dr. Penny respectively. They differ principally in the nature of the oxidizing solution employed.

MARGUÉRITE'S PROCESS.—When potassium permanganate is added to the solution of a ferrous salt, the latter is converted into a ferric salt according to the following equation for ferrous chloride :—

$$10FeCl_2 + 2KMnO_4 + 16HCl = 5Fe_2Cl_6 + 2KCl + 2MnCl_2 + 8H_2O.$$

Potassium permanganate solutions, even when very weak, have a decided red colour, which is immediately discharged as long as any ferrous salt remains unaltered; but immediately that point is reached, even a single drop in excess gives a characteristic rose tint to the solution, so that the exact termination of the operation is easily seen.

PENNY'S PROCESS.—Ferrous salts in solution containing free acid are converted into the corresponding ferric salts when treated in the cold with a solution of potassium bichromate. The following is the reaction for ferrous chloride :—

$$6FeCl_2 + K_2Cr_2O_7 + 14HCl = 3Fe_2Cl_6 + 2KCl + Cr_2Cl_6 + 7H_2O.$$

The change of colour in the solution by this method is not sufficiently marked to show the exact termination of the reaction, and a secondary mode of indication is therefore required. For this purpose a weak solution of potassium ferricyanide (red prussiate) is used. A slab of white porcelain is covered with dots of this solution, and a drop taken by a stirring-rod from the assay liquor is added to a fresh dot at intervals during the addition of the standard solution. So long as any notable proportion of ferrous salt remains, a deep blue colour is immediately produced; but as the oxidation proceeds, the blue tint becomes paler and forms more slowly, and when the oxidation is complete, no change of colour takes place.

*Preparation of Standard Solutions.*—In order to avoid calculations as much as possible, standard assay solutions are made of such strength that a unit volume measured in the burette stands in some simple relation to a unit weight in the assay sample. For instance, for metric measures one cubic centimetre may represent one centigramme of pure iron, or one per cent. upon an assay weighing one gramme; and as the burette can be read to less than $\frac{1}{10}$ of a centimetre, the result will be accurate to within $\frac{1}{10}$ per cent. Solutions of this strength are of the following composition :—

| | Grammes per litre. | Grains per gallon. |
|---|---|---|
| 1. Potassium bichromate | 2·821 | 197·470 |
| 2. Potassium permanganate | 8·764 | 613·480 |

In either case, the weighed quantity of pure crystallized salt may be dissolved in somewhat less than the proper quantity of water, and diluted to the right strength when the actual value or standard has been determined by experiment. This is done by operating upon a weighed quantity either of iron of known composition, or preferably of pure ferrous sulphate, or of ammonio-ferrous sulphate. The latter salt is perhaps the most convenient, as it may easily be preserved without alteration, and contains exactly one-seventh of its weight of iron.

The bichromate solution being quite stable when protected against loss by evaporation, may be prepared and kept in large quantities, but the permanganate is more susceptible to alteration, and should not be kept too long. It is also immediately decomposed by contact with organic matter, and therefore must be kept in glass-stoppered bottles.

*Solution of the Ore.*—About one gramme of the finely powdered sample is heated in a hard glass flask with strong hydrochloric acid until the whole of the iron is dissolved. The insoluble residue, when light-coloured, consists only of silica and clay, and need not be separated; but when it contains carbonaceous matter, as is the case with blackband ironstone, the solution must be filtered. The dissolved iron, being wholly or partly in the state of ferric chloride, is next reduced to the ferrous state. For this purpose the solution is diluted and treated with zinc or sodium sulphite. In the first case, it is necessary to use zinc free from iron in tolerably large granulated pieces; the small trace of lead usually present is of no consequence. The liquor is boiled until it is completely decolorized, and when cool is poured off into a porcelain dish, leaving the undissolved lumps of zinc behind. These are rinsed with water and the washings added to the original solution. The treatment with sodium sulphite is similar; it is added either in solution or crystals, and the decolorized liquor must be boiled until it ceases to smell of sulphurous acid. The use of zinc, though convenient, is inadmissible when the ore contains titanium, as the higher titanic chloride, $TiCl_4$, is reduced by it to titanous chloride, $Ti_2Cl_6$, which being re-converted by the standard solution, will cause the amount of iron to appear higher than it really is. Another very accurate method of reduction is to pass sulphuretted hydrogen through the solution until it is decolorized, and then boiling until the excess of gas is completely expelled.

*Determination of the Iron.*—The assay solution when diluted and transferred to a porcelain dish, as described above, is placed beneath the spout of the burette, and the standard solution is added in quantities of two to three centimetres at a time, the contents of the basin being carefully stirred after each addition. The quantity added each time is diminished as the operation proceeds, and finally is made by single drops. The slightest excess of permanganate gives a pink colour to the liquor, so that the exact termination is readily determined. With the bichromate this point is not quite so easily seen, and it is therefore necessary to repeat the testing with the red prussiate indicator at short intervals when the standard solution ceases to produce any decided change of colour in the assay.

When permanganate is used, it is better to convert the ferrous salt in the assay into a sulphate, which may be done by evaporating the hydrochloric acid solution nearly to dryness, adding sulphuric acid, and boiling until the hydrochloric acid is driven off, as under certain conditions permanganate is decomposed by hydrochloric acid when concentrated.

As has already been stated, the permanganate method is inaccurate when the ore contains organic matter, and for this reason the use of indiarubber tubes in connection with the burette must be avoided, a glass stopcock being substituted.

The state of oxidation of the iron in the ore may, when the total iron has been determined by the method described above, be found by making a second assay, omitting the reduction by zinc or by sulphurous acid. In this case only the iron originally existing as ferrous oxide is determined, and from the difference of the two results that contained as ferric oxide is computed. Special care must be taken to prevent the access of air or oxygen to the flask during solution. This may be done by dissolving the iron in a partially closed tube, or adding some carbonate of sodium to the liquor to expel the air by the carbon dioxide given off.

Penny's process is generally to be preferred to Marguérite's, for the following reasons :—

1. The solution of potassium bichromate is less liable to decomposition by long keeping than that of permanganate, and requires less frequent standardization.

2. The result obtained is more reliable, since it is not subject to error through evolution of chlorine.

3. It occupies less time, as filtration is not generally necessary, even when carbon or other organic matter is present, and no time is lost in waiting for the solution to cool, as is the case with permanganate.

*Dry and Wet Assay.—Comparative Yields.*—As the result of dry assay is *cast-iron*, a substance sometimes containing above 15 per cent. of other elements, while the wet assay expresses the amount of *pure iron* in the ore, the results obtained by the former method should, in all cases, indicate a higher yield than those obtained by wet assay from the same ironstone.

The following series of dry and wet assays, carefully made in Dr. Percy's laboratory, will serve to show the usual amount of difference between the results obtained :—

| | 1. | 2. | 3. | 4. | 5. | 6. |
|---|---|---|---|---|---|---|
| Iron by dry assay . | 73·30 | 70·30 | 59·60 | 35·30 | 42·10 | 34·30 |
| „ wet „ . | 69·75 | 68·08 | 57·57 | 33·35 | 37·55 | 32·13 |

The wet assays were made by means of a standardized solution of potassium bichromate or of permanganate, and the dry assays in brasqued crucibles by the Swedish method.

## ANALYSIS OF IRON ORES.

The complete analysis of an iron ore is an operation which is both tedious and difficult, involving numerous precipitations, filtrations, washings, dryings, and weighings, and should only be attempted by a chemist having considerable experience of mineral analysis. The following is an outline of the more important processes employed for the systematic analysis of ores of this class, but the reader who may require more comprehensive information on this subject is referred to a 'Memoir on the Iron Ores of Great Britain,' founded on investigations conducted by Messrs. Dick & Spiller in the laboratory of the Royal School of Mines, published, in parts, between the years 1856–62. He will also do well to consult an admirable paper by Mr. E. Riley, in the 'Quarterly Journal of the Chemical Society,' vol. xii. p. 13.[1]

*Water.*—The amount of *hygroscopic water* present in an ore is determined by exposing a weighed quantity in a finely powdered state to a temperature of 100° C. in a water-bath until it ceases to lose weight. *Combined water* is estimated by heating the dried ore, or a weighed quantity of the undried ore, to redness in a hard glass bulb, to which is adapted a chloride of calcium tube, of which the weight has been previously ascertained. By this treatment the water and other volatile matters are expelled, the water only being retained by the calcium chloride, so that its amount may be found directly by re-weighing the tube. When undried ore is operated on, the amount of water lost by exposure to 100° C. must be subtracted from the increase in weight of the chloride of calcium tube, in calculating the percentage of combined water present.

*Attack by Hydrochloric Acid, &c.*—A weighed quantity of the finely pulverized ore is digested in strong hydrochloric acid until no further action takes place, and, after boiling for an additional ten minutes, the solution is diluted with distilled water and filtered. The *insoluble matter* which remains on the filter is well washed with distilled water, dried in a water-bath, ignited in a platinum dish, and weighed.

The filtrate from insoluble matter is now boiled with the addition of nitric acid or potassium chlorate, for the purpose of peroxidizing the iron, and after being rendered neutral with sodium carbonate, is boiled with an excess of sodium acetate and filtered hot. The precipitate thus obtained is washed with hot water, the filtrate received in a flask, rendered alkaline by ammonia, and a few drops of bromine added. After corking the flask, it is set aside for twenty-four hours, when it is boiled, and the precipitated hydrated manganic peroxide separated by filtration, washed, and converted by ignition into $Mn_3O_4$. From the weight of this the proportion of *manganese* is computed. The filtrate from the last operation may contain lime and magnesia; the former is precipitated by oxalate of ammonium, and may be weighed either as carbonate or as sulphate of calcium. Oxalate of calcium is converted into carbonate by ignition at a low red heat, and into sulphate by the addition of weak sulphuric acid and heating until the excess of acid has been driven off. The most accurate results are perhaps obtained by the latter process, although, when the carbonate obtained by igniting the oxalate is subsequently heated with a little carbonate of ammonium until all traces of the volatile salt have been expelled, the former method is sufficiently exact. From the amount of the calcium salt thus found the percentage of *lime* present is calculated. To the filtrate from the oxalate of calcium, sodium phosphate is added, and the solution is set aside for twenty-four hours, during which time, if magnesia be present, a crystalline double phosphate of magnesium and ammonium will be deposited. This, on being heated, gives off water and ammonia, and is converted into magnesium pyrophosphate, $Mg_2P_2O_7$, containing 36·33 per cent. of *magnesia*.

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[1] **'On the General Occurrence of Titanic Acid in Clays, and the method employed to estimate it; on the Analysis of Iron Ores and Siliceous Minerals containing Iron; the separation of Oxide of Iron from Titanic Acid; and the methods of estimating Iron.' Also, 'The Assay and Analysis of Iron and Steel,' by Thomas Bayley. 1884.**

The precipitate produced in the hydrochloric solution by boiling with excess of sodium acetate, consisting of basic acetates of iron and aluminium with phosphoric acid, is dissolved in hydrochloric acid and boiled with excess of caustic potash in a platinum dish. Ferric oxide will be thus thrown down, while the alumina at first precipitated is re-dissolved, and may be separated by filtration. The filtrate is acidified by hydrochloric acid and boiled with the addition of potassium chlorate, for the purpose of destroying any soluble organic matter due to the action of the caustic alkali on the filter, rendered nearly neutral by ammonia, and finally made alkaline by carbonate of ammonium. The alumina, together with phosphoric anhydride, will, when that substance is contained in the ore, be now precipitated, and must be washed, ignited, and weighed.

The amount of $P_2O_5$ present is subsequently ascertained by the method given on p. 166, and its weight is deducted from the former weighing. The difference gives the amount of *alumina* in the ore.

The precipitated ferric hydrate remaining after the separation of alumina is re-dissolved in hydrochloric or sulphuric acid, and the amount of *iron* determined by a standard solution of either bichromate of potassium or of the permanganate.[1]

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[1] Chemists have now generally given up the use of potash in the analysis of iron ores; ferric oxide, alumina, and phosphoric anhydride are weighed together, and the mixture then dissolved in hydrochloric acid. Any silica it may contain is separated by filtration, the iron is determined by a standard solution, and the corresponding amount of ferric oxide is deducted from the weight of the mixed precipitate; phosphoric anhydride is estimated by a special determination, and its weight also deducted, the residue being alumina. Estimation by loss is generally objectionable, but experience has shown that this method is most reliable.

*Sulphur.*—Sulphur may exist in iron ores either as sulphates, soluble or insoluble in hydrochloric acid, or as insoluble iron pyrites. The sulphur present in the form of soluble sulphates may be determined by digesting a weighed quantity of the ore in dilute hydrochloric acid, filtering, and adding barium chloride to the filtrate. The sulphate of barium precipitated is thrown on a filter, washed, dried, ignited, and weighed, and from its weight is calculated the percentage amount of sulphur existing in the state of soluble sulphates. If insoluble sulphates are present, the amount of sulphur they contain may be determined by fusing the pulverized ore with carbonate of sodium, treating the fused mass with water, decanting, adding hydrochloric acid in excess to the solution, and precipitating by barium chloride as before directed. If sulphates soluble in hydrochloric acid are present at the same time with sulphates insoluble in that menstruum, both operations will be necessary for their separate determination.

The sulphur present as iron pyrites is best determined by fusing the substance in a finely divided state, with a mixture of nitre and pure sodium carbonate, in a gold crucible, dissolving in dilute hydrochloric acid, evaporating to dryness, and separating insoluble matter by re-attacking with hydrochloric acid and filtering. Sulphate of barium is precipitated from the filtrate on the addition of barium chloride.

*Phosphoric Anhydride.*—This may be determined by the method first proposed by Dick. A weighed quantity of the ore is dissolved in hydrochloric acid, the insoluble matter is separated by filtration, the iron in the filtrate reduced to the form of a protosalt by sodium sulphite, and all free sulphurous anhydride driven off by boiling; a small portion of the solution is oxidized by nitric acid and added to the remainder, which is then nearly neutralized with ammonia, acetate of sodium added, and the mixture boiled. All the phosphoric acid is precipitated, together with a small amount of ferric oxide, of which a portion goes down as ferric acetate. The precipitate is collected and dissolved in excess of hydrochloric acid; tartaric acid, ammonia, and a magnesium salt added. and the liquid allowed to stand at least twenty-four hours; the precipitate is collected on a filter, washed, dried, and ignited, and weighed as $Mg_2P_2O_7$, containing 63·67 per cent. of $P_2O_5$. For ores containing only minute quantities of phosphorus the method of precipitation by molybdic acid may be used.

*Carbonic Anhydride.*—The amount of $CO_2$ present is best determined by dissolving a known weight of ore in hydrochloric acid in a small flask provided with a safety-funnel, and collecting the gas evolved in potash-bulbs, after drying it by passing through a chloride of calcium tube.

*Indirect Determination of Manganese.*—A method of determining manganese in iron ores or metal, which is much more rapid than that previously given, has been introduced by Mr. Pattinson, who finds that manganese may be completely precipitated from a hydrochloric acid solution containing a certain proportion of ferric chloride, as hydrated dioxide, by a solution of bleaching-powder or bromine and an excess of carbonate of calcium, the temperature of the liquor being kept at about 60° to 70° C. The dark brown precipitate, when freed from chlorine or bromine by washing, is transferred in the filter to a beaker, in which a measured volume of a solution of ferrous sulphate in sulphuric acid containing a known quantity of iron has been placed. The precipitate is readily dissolved, the manganese dioxide producing manganous sulphate, and oxidizing an equivalent proportion of ferrous to ferric sulphate. The residual amount of ferrous sulphate is then determined by bichromate standard solution, as in iron assaying, and the difference between this and the total quantity taken gives the amount peroxidized by the oxygen liberated from the manganese dioxide in dissolving, and from it the metal in the latter is computed according to the following equation:—

$$2FeOSO_3 + MnO_2 + 2SO_3 = MnOSO_3 + Fe_2O_33SO_3,$$

each unit of ferrous sulphate oxidized representing 0·18 unit of manganese. In order to compensate for any error due to the reducing action of the paper of the filter, a filter of similar size should be added to the acid ferrous sulphate liquor before determining its standard. This method is sufficiently accurate for most purposes when the proper precautions[1] are taken; but to ensure complete precipitation, in the first instance, it is necessary that the original solution should contain, at least, half as much iron as manganese.

*Titanic Oxide.*—The amount of titanic oxide present in ordinary iron ores is usually so small that its determination is not of much commercial importance. For the processes employed for the exact estimation of this substance, which are somewhat complicated, the reader is referred to the memoir before referred to of Mr. Riley, who has devoted much careful attention to the metallurgy of iron. It is, however, a mistake to suppose that titanic oxide is left with the silica in the analysis of iron ores, or that it is completely separated by evaporation to dryness; a considerable amount is dissolved by strong hydrochloric acid. In iron ores containing from 20 to 30 per cent. of titanic oxide it has been found that nearly the whole had been dissolved by hydrochloric acid, and that only a very small amount remained with the silica.

*Insoluble Residue.*—The insoluble residue from the attack of hydrochloric acid is, for commercial purposes, not usually examined, and is generally returned as "insoluble siliceous matter." With a view to ascertaining its influence on the working of the ore, it is, however, sometimes desirable to determine its exact composition, in which case it may be fused in a platinum crucible with four times its weight of an equal mixture of the carbonates of potassium and sodium, and subsequently treated with hot water, which dissolves out alkaline sulphates, if present, together with alkaline silicates. The residue, after decanting the aqueous solution and washing, is treated with hydrochloric acid and evaporated to dryness in the usual way; the filtrate is added to the liquid obtained by evaporating the aqueous solution to dryness with addition of hydrochloric acid, re-treating with hydrochloric acid and filtering. Should insoluble sulphate of barium or sulphate of strontium have been present in the ore, a precipitate will be at once formed, and the insoluble sulphates are separated by filtration; in the filtrate will be found oxide of iron, alumina, lime, magnesia, &c., which may be separated and estimated by processes already described.

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[1] For further details, see Mr. Pattinson's paper, 'Journal of the Iron and Steel Institute,' 1879, p. 209.

## METALLURGY OF IRON.

As has been previously stated, iron is used in the three principal states of cast-iron, steel, and wrought-iron, which differ from each other mainly in their amount of contained carbon, and it is to the production of these three substances that the iron-smelter's operations are directed. Taking the ore as a point of departure, any one of the three products may be obtained from it, either by a single furnace operation, essentially one of reduction by gaseous or solid carbon, or by a combination of two or more alternating operations of reduction, oxidation, or *fining*, and carburizing or *cementation*. The former are known as *direct* and the latter as *indirect* processes.

The direct production of malleable iron will be first considered, as representing the simplest and most ancient process, as well as that from which all the more complex methods have been developed.

## DIRECT PREPARATION OF MALLEABLE IRON.

When a lump of pure and easily reducible iron ore, such as soft red hæmatite, or a brown iron ore formed by the oxidation of spathic ore, is heated in a bed of ignited charcoal in a smelting fire, it is readily reduced, and a mass of metal obtained similar in shape to that of the ore treated; if the lump is sufficiently large, one end may be reduced and drawn out to a bar or rod, while the other remains as unaltered ore. The reducing agent is carbonic oxide formed by the action of the blast upon the fuel in excess, and the operation exactly represents the essential features of the primitive native methods of iron-smelting practised in forges by the inhabitants of India and Africa, and the Catalan and Corsican processes, which have survived until recently in Europe.

Dense and impermeable ores, such as rich magnetite and specular hæmatite in lumps, are unfit for the direct process, and must be pulverized in order to provide sufficient surface for the action of the reducing gases, and added by small quantities at a time to the bed of ignited charcoal. This condition is characteristic of the German blooming process, which, though now extinct in Europe, is still practised upon a considerable scale in the United States of America.

If, instead of bringing the ore directly into contact with the fuel, the latter be converted into carbonic oxide and passed through the ore in a chamber or retort, which may or may not be heated externally, the condition of reduction is similar to that described pp. 179–183, for the production of sponge iron by carbonic oxide. This method is represented in the direct processes of Clay, Chenot, Blair, Siemens, &c.

NATIVE FORGE OR BLOOMING PROCESSES.—Very primitive methods of iron-making are still practised in many localities in India, Burmah, and the East Indian Islands. The furnaces used are of two kinds, one being worked with an artificial blast and the other without. These latter probably represent an older class, used before the discovery of bellows; at least such is believed to have been the case in Europe, as remains of furnaces have been found in many places on hill-sides exposed to the wind, but far removed from watercourses whence power for moving blowing engines or hammers could be obtained. These furnaces are known as *Windöfen* in Germany, and as *bayles* or *boles* in the North of England, where they were employed for smelting lead ores. A Burmese furnace of this class is described by Mr. W. T. Blanford[1] as having a stalk about 10 feet high cut in a bank of sandy clay, with a solid wall in front of about 3 feet thick. The horizontal section is trapezoidal with the longer side in front, the breadth increasing from 1¾ foot at the top to 5½ or 6 feet at the bottom, and the depth from front to back increasing from 21 inches at the top to 2 feet at the widest part about midway down, and diminishing to 1 foot at the bottom. An arch cut in the foot of the bank communicates with the rectangular aperture at the bottom of the stack, of the full width of the latter and about a foot high. When the furnace is at work this aperture is fitted with about 20 nozzles or tuyers of burnt clay, the intermediate spaces being plastered up with soft clay. When done, a lighted brand is placed at the bottom of the stack, and a bed of charcoal is formed, upon which ore and charcoal are charged alternately in the total proportion of about 420 lbs. of the former to 437 lbs. of the latter. In about eight or nine hours after lighting, a hole about four inches square is cut through the side of the stack next the bottom to allow the slag that has accumulated to escape, after which it is stopped and re-opened at intervals of about half an hour until the flow of slag ceases. The furnace is then allowed to cool and the clay stopping at the bottom is broken out to allow the reduced iron, which forms a long narrow mass of the shape of the hearth, to be removed. This weighs from 90 to 120 lbs., and consists of reduced iron mixed with slag and unconsumed charcoal, from which it must be freed by re-heating in order to obtain a compact bar.

In Bengal and the Northern and Central Provinces of India, furnaces blown by bellows are in general use. The furnaces are made of clay, usually alluvial mud of a very sandy character, and have stacks of a slightly conical form, from 3 to 5 feet high and from 18 to 20 inches diameter, with an arch cut out at bottom for the passage of the blast nozzle, which is stopped while the furnace is at work. A platform with a raised border with an upward slope projects from the furnace top about 3 feet. This is supported upon a wooden framing, and upon it the ore and fuel are laid before being charged into the furnace. The bellows in use are of several kinds; a common form has a cup-shaped leather with a central aperture connecting a low cylindrical box with a nozzle brought through one side. The box is filled with air by the action of a bamboo spring which lifts the leather cover, and is emptied by pressing it down with the heel. Two such bellows are provided for each furnace, which are worked by one man, who stands upon the leather cover, which rises and falls alternately as the heel is raised or depressed. Both bellows blow into one tuyer. The ores smelted are chiefly brown hæmatites of various degrees of purity; generally they are broken small, and previously calcined with charcoal or brushwood. The manipulation is very similar to that previously described. In working, the smelters are very particular in maintaining a flame at the furnace top to prevent the escape of unconsumed carbonic oxide on account of its poisonous character. The operation lasts from four to six hours, the slag being tapped off at intervals in the usual way. The reduced metal may either be taken out of the bottom or preferably lifted out by a pair of tongs from the top, which leaves the furnace in working condition, and gives a firmly coherent mass of iron, weighing from 20 to 40 lbs.

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[1] Percy, 'Iron and Steel,' p. 271.

The natives of Borneo prepare their iron from clay ironstone, which is treated in a furnace built of yellow clay, and tied round by hoops of bamboo. Its height is a little more than 3 feet, and its external diameter nearly 10 feet, the thickness of the walls being 2 feet; it is square on the inside, narrowing towards the bottom to a rectangular hearth, 2 feet long by 1 foot 7 inches wide. Each furnace has three clay tuyers with an opening for running off the slags, and an external basin for their reception. The blowing-machine is a single-acting cylinder of wood, open at top and closed at bottom, the blast being conveyed from the lower end to the tuyer by means of bamboo tubes. The piston is packed with feathers, and the piston-rod is attached to a long bamboo, which, acting as a spring, brings it back again when pressed down to the bottom of the wooden cylinder.

The ore, preparatory to smelting, is interstratified with wood and roasted in heaps, and, after being broken into pieces of the size of nuts, is mixed with ten times its bulk of charcoal, and charged into the furnace. When it has been two-thirds filled with charcoal, the mixture of ore and fuel is added in sufficient quantity to form a conical heap above its mouth.

The piston is worked at the rate of about forty strokes per minute; the slag is tapped off every twenty minutes, and a lump of iron, weighing about 100 lbs., is finally obtained. This is taken out at the bottom of the furnace by means of wooden tongs, and is removed to a bed of slag, where it is worked by wooden mallets into the shape of roughly formed parallelopipedons; such a mass is the result of the labour of four men during one day. The mass retains much intermingled slag, which is removed by dividing it into ten pieces, which are hammered out into bars suitable for making sword-blades, during which operation a loss of 25 per cent. in weight is experienced. Soft or steely iron may be produced at pleasure according to the nature of the fuel employed, and the proportions of the charge.

**The Catalan Forge.**—Both on the French and Spanish sides of the Pyrenees the method of obtaining malleable iron directly from the ore has been practised until recent times in a furnace known as the Catalan forge. This is a low hearth or open fire, whose top is level with the floor of the smithy or shop, which receives a blast from an inclined tuyer above it, as shown in figs. 35, 36. The hearth, about 30 inches deep, is enclosed by four walls; that on the side of the tuyer, *c*, known as the *porges*, is upright, and built of heavy blocks of cast-iron; the opposite side, *d*, called the *ore*, is also of cast-iron blocks, which are curved outward, so that the breadth of the furnace increases from about 2 feet at the bottom to nearly 3 feet at the top. The third wall, *a*, or *chio*, about 20 inches high, also upright and of cast-iron, has a notch in it at the bottom for running out the slag. The fourth side, or *cave*, *b*, is built of rough masonry with a slight upward batter. The hearth bottom is made of a block of sandstone slightly hollowed at the top. The tuyer, N, is made of sheet-copper lapped round without soldering. It rests on the top of the porges, and plunges at an angle which may be varied with the requirements of the work; the proper manage-

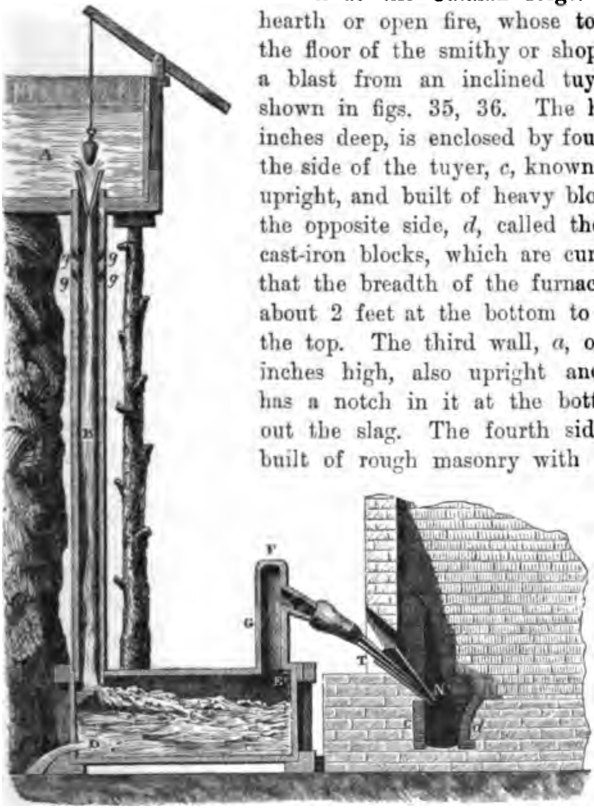

Fig. 35.—Catalan Forge and Trompe; vertical section.

ment of the blast being one of the niceties of the process. Usually the angle is from 35° to 40° from the horizontal plane.

These forges are almost invariably placed on the declivity of a hill, and are supplied with air by a water blowing machine, called a *trompe*. This consists of a large cistern, A, which is supplied with a constant stream of water, and connected with the box, C, by two wooden pipes, B, each about 20 ft. in length.

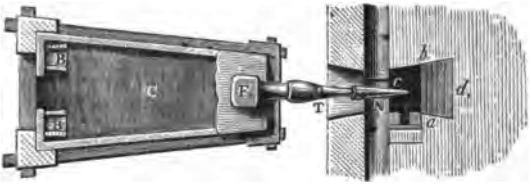

Fig. 36.—Catalan Forge and Trompe; plan, partly in section.

The lower box, C, which is firmly secured on all sides, and closely united to the pipes, B, is pierced with two openings, the one, D, near the bottom for the escape of the water, and another in the lid, at E, through which the air escapes into the furnace through the tube, G, F, and the nozzle, T, which are connected by a leathern hose.

The openings of the pipes, B, are, at their point of junction with the reservoir, partially closed by a sort of wooden funnel, which causes the water to descend in the middle portions of the upright pipes, instead of adhering to and running over their inner surfaces, as it would be otherwise liable to do. A little beneath the openings of these funnels, called the *étranguillons*, small openings, *g*, are cut in an inclined direction through each tube; these are called the *aspirateurs*, and serve for the passage of the air drawn into the apparatus by the downward motion of the stream of water. The two upright pipes, B, are firmly secured into the lid of the lower box, C, and are placed immediately over, and a short distance above a wooden shelf, on which the descending currents of water are, by their fall, broken into foam.

The action of the apparatus may be explained as follows: the water flowing from the upper basin, A, draws down with it a current of air, which enters through the holes, *g*, in the vertical pipes, B, and passes into the lower cistern, C. The water which is broken by its fall on the bench below, escapes by the opening, D, whilst the air which has been drawn with it into the lower box, escapes by the aperture, E. The position of the boards constituting the étranguillons is easily regulated by means of wedges, which allow of the descent of a larger or smaller supply of water, according to the requirements of the trompe. In order, during the working of the machine, to regulate the amount of air passing into the furnace at the different stages of the operation, each of the descending pipes is provided with a plug, suspended by a lever and iron rod, by means of which the current of water, and consequently that of air, is readily controlled by the workmen. The air is delivered at a pressure of 1½ to 2¾ inches of mercury.

The hammer employed for forging the iron produced has a cast-iron head, and weighs from 12 to 14 cwts., with a face about 16 inches long and 2⅛ inches broad. The wooden helve, frequently made of beech, is hooped with iron, and carries a trunnion-ring dividing it into two unequal arms, the longer one being on the side of the anvil. The hammer makes from 100 to 150 blows per minute, and is raised by a series of cams, arranged around the axle of a water-wheel, and acting on the tail of the shorter arm. The anvil is composed of a block of steely iron, fastened by a tenon, on a large mass of cast-iron, which is itself securely bedded either on wooden piles or on a heavy block of stone, sunk beneath the floor of the foundry.

In order to understand the method of working this forge, let us suppose that a mass of iron or bloom has been just extracted from the furnace, and that the workmen are ready to clean out the hearth for the purpose of commencing another operation.

To do this they first remove from the hearth the burning charcoal which it contains, and then carefully scrape off from the sides any portions of scoriæ, or other fused matter, which may be adhering to them. They then throw burning charcoal into the hearth, which they subsequently fill with this fuel up to the level of the tuyer. The hearth is now divided, either by a shovel or by a piece of sheet-iron, into two compartments parallel to the face of the porges, and in such a way that the distance between the porges and the shovel may be twice as great as that comprised between it and the "ore" or *contrevent.* Charcoal is now added in the space between the shovel and the tuyer, and on the opposite side is piled the ore reduced to pieces about the size of eggs. The shovel is successively raised in proportion as the space is filled up, and in this way a saddle-backed heap, *a*, *b*, *c*, fig. 37, is raised against the contrevent, which is terminated in one direction by the side called the chio, and in the other by the face of the cave. The surface, *a*, *b*, is now covered with damp charcoal-powder, and the space, A, between the heap of mineral and the porges, is entirely filled with fresh charcoal, in pieces of moderate size.

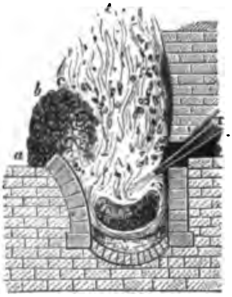

Fig. 37.

When the hearth has been thus prepared, the trompe is set in action, and the blast admitted. This is at first done with considerable caution, but the blast is progressively increased until it is allowed to play into the fire at its full pressure. Whilst this is going on, the heap of broken ore is gradually roasted and reduced, and the workmen, taking advantage of this opportunity, forge into bars the mass of iron produced by a former operation, which for this purpose is finally divided into four separate pieces, or *massouquettes.* These fragments are placed in the midst of the mass of charcoal lying between the heap of ore to be reduced and the nozzle which furnishes the air necessary for carrying on the combustion of the fuel, and, after being duly heated, they are placed under the hammer, by which they are made to assume the required form.

As the operation advances, and the fuel is consumed, fresh charcoal is added to supply its place, and powdered mineral, obtained by sifting the ore as it comes from the mine, is occasionally sprinkled over the surface of the fire. These siftings, which are called *greillade*, are slightly moistened with water, after being thrown on the hearth, as they would otherwise be liable to be blown away by the force of the blast, and have a tendency to pass too rapidly towards the bottom of the fire, through the interstices occurring between the fragments of the fuel.

The charcoal in the immediate neighbourhood of the tuyer, which is subjected to the full action of the blast, is rapidly consumed with the formation of carbonic oxide. This, from the construction of the furnace, has to pass through the openings left between the lumps of mineral, before finding its way into the open air; the mineral, which has now lost all traces of its volatile constituents, and is very strongly heated, is in a great measure reduced by this means to the state of spongy metallic iron, while the carbonic oxide is at the same time converted into carbonic anhydride, and escapes in that form into the atmosphere. Another portion of the oxide of iron present, instead of being obtained in the metallic state merely becomes converted into protoxide, which, uniting with the siliceous matters of the charge, gives rise to a large quantity of very liquid slag, which accumulates on the bottom of the hearth, and is occasionally drawn off by the hole in the face of the chio.

At the expiration of two hours from the commencement of the operation, the full blast of the blowing-machine is admitted to the furnace, and the greillade, which constantly descends with the fuel, begins to furnish a certain quantity of slag and spongy iron, which accumulate at the bottom of the hearth. At this stage of the process, the founder begins to prepare for the formation of the *massé* or *bloom*, and, by passing an iron bar between the contrevent and roasted mineral, pushes forward those portions of it which he judges to be in the most forward state in the direction of the nozzle by which the blast is admitted. Fresh additions of charcoal and greillade are also successively made during the whole period of the operation, and at the expiration of about five hours from the time of its commencement, the entire charge has reached the bottom of the furnace, where the spongy iron is collected by the workmen with a long iron scraper, and formed into a bloom; this is afterwards carried to the hammer, by which the slag is expressed, and its particles closely welded together in a compact form. When the bloom has by this means been welded into a solid mass, it is again put under the hammer, and cut by a kind of heavy steel knife into two equal portions, called *massoques*, which after being a second time heated in the furnace, are made to assume the form of elongated prisms. Each of these is subsequently divided, by a blow of the hammer on the back of the cutter, into two equal parts or massouquettes, which are drawn out into bars during the first period of the succeeding operation.

Each charge requires six hours for its conversion into malleable iron, but during the last hour of fusion, such of the labourers as are not otherwise engaged are occupied in breaking the ores ready for the next operation, and sifting the greillade which is to be sprinkled on the surface of the fire.

In 1841 the weight of ore treated at each operation, in a hearth of the largest size, was about 9½ cwts., containing from 45 to 48 per cent. of iron; the fuel consumed averaged 10¾ cwts., and the produce of bar-iron 3 cwts. The consumption of material per 100 lbs. of bar-iron was, of ore 312 lbs., and charcoal 340 lbs.; the average cost of production was about £17, 5s. per ton.

In 1868, according to Mussy, the product of one operation weighed 170 kilogrammes, and the prime cost was as follows per 100 kilogrammes:—

| | Francs. |
|---|---|
| 300 kilogrammes ore at 15 francs per ton . . . . | 4·50 |
| 200 „ charcoal at 6 francs „ . . . . . | 18·56 |
| Wages . . . . . . . . . . . . | 4·72 |
| Maintenance of plant and management . . . . | 0·71 |
| | 28·49 |

Or about £11, 8s. per ton.

The metal obtained by this method consists of a variable mixture of ordinary and steely iron, the relative proportions of which are regulated by the way in which the furnace is worked; for if considerable inclination be given to the tuyer, and the siftings are plentifully thrown on the fire, the product is chiefly soft iron, whilst if the nozzle be more nearly horizontal and the greillade but sparingly supplied, a larger product of steely iron is the result.

AMERICAN BLOOMERY PROCESS.—In the northern parts of the States of New York and Vermont, adjacent to the Canadian frontier, and at a few other localities in New Jersey and in Canada, the direct method of making iron from the ore is still practised in a furnace very similar to that used for fining cast-iron in the so-called Walloon or German process. It differs from the Catalan forge by being nearly square in plan, while the bottom is a cast-iron plate cooled with water; the tuyer is also of cast-iron. The enclosed space, known as the fire-box, is about 30 inches square and 19 inches deep below the shelf or fire-plate on the working side, and 12 or 14 inches below the tuyer. The hearth is covered by a hood of brickwork, enclosing a series of three or more parallel syphon pipes, in which the blast is heated by the waste flame before it enters the tuyer. The ores treated are chiefly magnetic, and are either siftings from the larger lumps separated for the blast-furnace, or concentrates obtained by crushing and dressing rock with interspersed grains of magnetite, and as nearly as may be perfectly pure. The titaniferous sands of the Lower St. Lawrence Valley have also been used.

The working bottom of the furnace is made of charcoal-dust and the slag from the hammering of a preceding bloom. Soft wood charcoal is used, and the ore is charged upon it by shovelsful of about 35 lbs., from one to three being added at a time as soon as the temperature is sufficiently high. The reduced iron begins to form a ball or *loupe* on the hearth in about ten or fifteen minutes. The re-heating of the former bloom goes on in the same fire. The duration of one operation or heat is about three hours, when 4 to 12 cwts. of ore are treated, giving a loupe or ball of 300 to 400 lbs. This is forged or shingled under a hammer to a bloom about 7 inches square, which is subsequently reduced to 5 inches, and then cut up and finished to four "billets" each of 70 or 80 lbs. About two tons of billets are made in twenty-four hours from one fire, consuming about 300 to 350 bushels of charcoal. Two tons of dressed ore, of 65 per cent., give one ton of bloom, or a loss of 30 per cent. This process is still practised on a considerable scale in the Adirondack regions, where blooms of iron of great purity are made, principally for conversion into steel by the open-hearth process. About 60,000 tons of direct blooms are made annually in the United States.

Corsican Process.—A method of producing malleable iron from the ore, which differed in many ways from the Catalan, was formerly used in Corsica and on the Italian Mediterranean coast. The forge or fire, which was blown through a single inclined tuyer by the blast from a trompe, had only two permanent walls, one perforated by the tuyer, and the other at right angles to it, faced with an iron plate containing the slag-hole. A platform of masonry about 3 feet high carried the hearth bottom, which was a layer of charcoal-dust moulded to a semi-elliptical or horse-shoe-shaped basin, whose bottom was about 4½ inches below the tuyer nozzle. This basin was enclosed by a wall of lumps of iron ore (rich specular hæmatite from Elba), diminishing in size upwards. The interior of the basin was divided into two parts by a horse-shoe wall of large pieces of charcoal, which enclosed a smaller interior well about the tuyer, and a larger outer one between itself and the outside wall. This exterior space was further divided into boxes or cells by two radial ribs, projecting from the main wall of charcoal. The outer cells were filled with small ore and dust resulting from the breaking-up of the lumps calcined in previous operations, and the interior well, forming the fire-place proper, with large charcoal. The heap, which was about 30 inches high, and contained about 10 cwts. of ore, was covered with charcoal dust; a few live coals were placed before the tuyer, and a gentle blast given by the trompe. As soon as inflammable gas appeared at the top it was ignited, and a moderate firing was continued for about forty minutes, after which the blast was turned full on for about an hour more, when the first operation, or roasting-process, was completed. The heap was then let down by pulling away the lumps of ore supporting it at the bottom, and the contents were broken up. These, consisting of masses of agglomerated ore from the outer chambers, which were filled with diffused particles of malleable iron, and a cake of slag on the hearth bottom, were, together with the small charcoal, put aside for the second or fining process, while the larger pieces of charcoal were used over again. The lumps of ore forming the casing were rendered friable by heat, and thus prepared for breaking, as a preliminary to reduction. For the fining operation, the hearth was made up with charcoal-dust, carefully freed from earthy matter by washing, arranged on two inclined planes, so as to form a gutter below the tuyer, upon which a bed of charcoal was laid to support the blooms of the preceding operation while re-heating for conversion into bars. This required about two hours, during which time the small ore, charcoal-dust, and slag of the roasting operation were gradually charged into the middle of the hearth. These, together with the slags formed during the re-heating, accumulated in the bottom and were tapped off in front. The quantity of agglomerated ore required to form one ball was then charged in front of the tuyer in two portions, the second being added as soon as the first was melted down, the operation being assisted by adding small quantities of re-heating slag from time to time.

When the iron had become a coherent mass in the bottom, the decarburization was completed by the addition of a small quantity of hammer-scale and finely divided ore. The blast was then stopped, water was thrown into the hearth to chill the surface, and the ball removed by tongs. After beating it with a wooden mallet to free it from adherent slags, a staff or porter-bar was welded on, to be used as a handle in the subsequent re-heating and forging; the latter being done under a water tilt-hammer, as in the Catalan process. The entire operation was completed in twenty-four hours, four hours being required for roasting and agglomerating, and four hours for the refining and balling of each fifth of the agglomerated charge. The iron was of good quality though irregular in composition; it made very durable horse-shoes, and was therefore somewhat steely. The yield of the ore was only about 38½ per cent. out of the 65 per cent. of iron actually contained, and the consumption of fuel was about 8⅝ tons per ton of iron produced. The forges only worked during seven months of the year, and each made about 26 tons of bar-iron, at a cost of £19 to £20 per ton.

Brazilian Process.—In the central part of the province of Minas Geraes in Brazil, wrought-iron is made from the ore in a furnace called a *cadinho*, which is either derived from the German Stückofen, or has been introduced from Africa by some negro slave. It has a stack of circular or slightly elliptical section, 10 or 12 inches in diameter, and 3½ or 4 feet high. The blast, produced by a trompe, is introduced by a single iron tuyer, placed 8 inches above the bottom, and opposite to it a hole 1 foot square is cut in the wall for the removal of the reduced iron. This is plugged with charcoal-dust from the floor of the forge during the operation. Three stacks are built close together, and have a common casing of large stones of granite or quartzite, and there are two such blocks of three furnaces in each forge, but only one is used at a time. When the furnace is started, it is filled with charcoal, lighted, and allowed to burn until the stack is half empty, when finely powdered rich specular ore is added, in quantities not exceeding two pounds at a time, alternately with fresh fuel, the stack being kept quite full. The ore is moistened with water to prevent its being blown out by the blast. After blowing for 1½ or 1¾ hour the charging is stopped, the furnace burns down, and the material accumulated on the hearth is removed. This is an agglomerated mixture of metal and slag, containing from 75 to 83 per cent. of iron. The slag is very infusible owing to the small proportion of silica (4 to 5 per cent.) in the ore, and consists largely of imperfectly reduced mineral. It contains 81·3 per cent. of ferric oxide, or 57 per cent. of iron. By re-heating, from 15 to 17 lbs. of finished iron are obtained from one operation, or from 1¾ to 2½ cwts. from the three fires daily. The weight of fuel consumed is seven or eight times that of the iron produced, and the latter only represents 22 per cent. of the actual contents of the ore.

An improved furnace somewhat resembling the Catalan, known as the Italian forge, is also used to some small extent. It is charged with 1 cwt. of ore, in lumps the size of an egg, stacked against the front wall, while the space behind is filled with charcoal, on which the ball of the preceding operation is re-heated during the earlier stages of the reduction. Afterwards about an equal quantity of powdered ore is charged at intervals with fresh charcoal. The operation lasts 3½ or 4 hours. The ball, as in the preceding case, is much mixed with slag and unburnt charcoal, and gives 80 or 90 lbs. of iron when re-heated and finished. About 40 per cent. of the iron in the ore is reduced, with a consumption of charcoal equal to 5½ times the weight of the iron made.

STÜCKOFEN.—The Stückofen or high bloomery-furnace, which represents the largest development of the small stack-furnaces of India, was similar in shape to a blast-furnace proper (such as is used for making cast-iron), with a stack formed by the junction of two conical frustra of unequal slope, the base of the upper or steeper one being downwards, and that of the lower, or more obtuse one, upwards; the broadest part, or boshes, in the plane of contact being about 4 or 5 feet wide, and the total height 14 to 16 feet. Two tuyers were placed near the bottom of the lower cone, leaving a space or hearth below them in which the reduced metal accumulated. The working was very similar to that of the Indian furnace, charcoal and iron ore being charged alternately until the latter was smelted down to produce from 4 to 6 cwts. of iron, when the ball or bloom was withdrawn by an opening made in an arch at the bottom of the stack, which was plastered up during the smelting process. The slag, resulting from the fluxing of the earthy matter of the ore and the ash of the fuel, accumulated in the hearth, and was let off by a tap-hole from time to time. The composition of slag of this kind is, according to Karsten, as follows:—

| $SiO_2$. | FeO. | $Al_2O_3$. | CaO. | MgO. | MnO. | Total. |
|---|---|---|---|---|---|---|
| 29·1 | 51·7 | 4·3 | 2·6 | 9·2 | 2·9 | 99·8 |

Three blooms of the weight given above were produced in twenty-four hours. This process, although common at one time in Europe, is now almost extinct. It was practised in the Eastern Alps and Thuringia, and more recently in the Lower Danubian territories and Finland. The blooms from the last-mentioned country are known as "Ösmund" iron, and the furnace in which they are made is more like an enlarged hearth than a Stückofen proper.[1]

CLAY'S PROCESS.—A patent was granted to William Neale Clay in 1837, and another in 1840, for processes for the manufacture of iron by welding together the crude spongy metal obtained by heating rich hæmatite with powdered charcoal. The method employed was to crush the better kinds of red hæmatite into lumps not larger than a walnut, and these, mixed with one-fifth of their weight of charcoal, coke, coal-slack, or any other carbonaceous materials, were subjected to a bright-red heat in a clay retort or other suitable vessel, until the iron was reduced to the metallic state. As soon as the reduction was considered complete, the spongy mass was transferred directly to a puddling-furnace, either with or without a further addition of coke; it was then balled in the usual way, worked into blooms under a tilt-hammer, and afterwards rolled into bars. Experiments by this process were first made on a small scale near Glasgow, and afterwards on a more extensive one in the vicinity of Liverpool; commercially it proved a complete failure, although iron of fair quality was sometimes produced. The iron was, however, frequently red-short; the chief cause of failure was the length of time required for reducing the ore, and the consequent heavy expenditure of fuel and labour.

It was subsequently attempted to effect the reduction of the ore directly in the puddling-furnace, but here again waste, in the shape of cinder, added to a large expenditure of time, fuel, and labour, caused the process to be abandoned for one in which a mixture of pig-iron, crushed hæmatite, and carbonaceous matter was treated in the puddling-furnace. The bar-iron thus produced was tolerably uniform in character, and of fair quality, fetching about the same price in the market as the better descriptions of Welsh bars. Upwards of 1,000 tons of iron were made by this process, but at such a heavy pecuniary loss as to cause it to be ultimately abandoned. Dr. Percy remarks of this process, that although it was not successful, it is important that the results obtained should be placed on record.

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[1] Percy, 'Iron and Steel,' p. 330.

CHENOT'S PROCESS.—Specimens of *éponges métalliques* or metallic sponges, obtained by the reduction of iron ore in retorts, were exhibited by M. Chenot at the first International Exhibition of 1851 in London, together with iron and steel made therefrom, and works were erected for carrying out the process near Bilbao, Spain, in 1852; at Charleroi, Belgium, in 1856; at Pontcharra, Isère, in 1856; and at Hautmont, Nord, in 1857. But of these, only the first continued in operation for any lengthened period. A modification of the process was for some time employed by the Messrs. Wright of Mostyn for the conversion of purple ore, resulting from the humid treatment of copper ores, into puddled bars, but the results were, commercially, unsatisfactory.

By this process the ore, if in mass, is broken into lumps of about 30 cubic centimetres, or 1½ cubic inch, but, if pulverulent, as in the case of oolitic ores, it is agglutinated, in certain cases with the addition of some reducing matter, such as resin, of which about 3 per cent. is added. Thus prepared, it is mixed with from 1¼ to 1½ its bulk, or, by weight, with 19 per cent. of wood-charcoal.

The furnace in which the reduction is effected consists of a cubical mass of masonry surmounted by a truncated cone of elliptical section. Two galleries, at right angles, traverse the supporting masonry, leaving four pillars at the angles, upon which are arches that support the fire-places. Within are constructed two vertical rectangular chambers or retorts, 2 m. long. 0·50 m. wide, and 8·50 m. high; at the bottom and below the level of the ground is a pit for the reception of the apparatus for discharging. The retorts slightly but gradually widen from the top towards the bottom, in order to facilitate the descent of the reduced charge; around each of these retorts is a series of vertical flues communicating, at bottom, with the fire-places, and, at top, with a flue opening into the air. The whole is firmly braced with bar-iron, and the bricks are made with tongues and grooves in order to prevent displacement and leakage.

If the reduced iron were exposed to the air while still hot, it would take fire, and again be converted into oxide. In order to prevent this, a *refroidissoir*, or cooler, of sheet-iron is placed below each retort, and beneath this there is another iron case into which the metallic sponge is discharged when sufficiently cooled. Below the latter, and on a level with the ground, is a waggon running on rails.

The charge of a furnace with one retort is 1,500 kilos., about 1½ ton, of calcined iron ore, and 500 kilos., ½ ton, of wood-charcoal; reduction is complete at the expiration of three days, when the freshly formed sponge is allowed to fall into the cooler, where it remains three days. The entire elaboration, including reduction and cooling, thus occupies six days. When perfectly reduced, the iron sponge has a light-grey colour, is soft, and can be easily cut into thin slices with a knife; it may be ignited by a match, when it continues to burn until the whole is completely oxidized. Under a pressure, stated at 3,000 atmospheres, this sponge has been compressed to one-fifth its original bulk. Great heat is evolved during the process of compression.

The process, as above described, is known as the external or indirect method of heating; by what is termed the internal or direct method, the reduction of the ore, instead of being effected by solid carbonaceous matter, is produced by a heated current of carbonic oxide gas. In this case, the reducing chamber is connected with gas-generators of the usual construction, by means of which a gaseous current, rich in carbonic oxide, is maintained upwards through the column of ore, and burns at top with its characteristic blue flame. A modification of this direct method of reduction, which was introduced by M. Tourangin, is said to have been attended with considerable economy in the cost of constructing the furnace.

The balling of the sponge is effected in a charcoal hearth similar to that employed for the manufacture of iron for tin-plates in South Wales. During the hammering of the ball, jets of blue flame escape from it in all directions, and the blooms may be rolled without re-heating into bars 1¾ inch square; these are cut into lengths, made into piles, re-heated, and rolled into merchant-bars.

Blair's Process.—A modification of Chenot's process was introduced by Mr. T. S. Blair at Glenwood, near Pittsburg, in 1870, the waste of fuel due to the difficulty with which the heat penetrated the contents of the retorts in the former process being obviated by applying fire to an annular space containing a mixture of ore and charcoal. The reduction was effected by gas produced from coke in a special gas-generator, the heat developed in the production of the gas being utilized in the furnace, so that no external firing was necessary. By means of a central tube on the top of each retort, about 6 feet long and 2 feet wide, an empty annular space of about 6 inches wide was enclosed. The retorts were heated externally by gas-jets, and the gas arising from the ore under reduction was fired at the top of the pipe, producing a body of flame that thoroughly heated the annular space through which the mixture of ore and charcoal was fed into the zone where active reduction took place; and as the thickness of the layer of materials was less than 6 inches, instead of 15 inches as in the Chenot retorts, they were much more readily heated. The height of the retort varied from 21 to 45 feet, and the diameter from 3 to 4 feet. The sheet-iron tube at the bottom was surrounded with a water-jacket in order to expedite the cooling of the sponge-iron. About 27 cwts. of coal and 8 cwts. of charcoal were consumed per ton of sponge-iron made; the loss of iron in the slags was from 14 to 16 per cent. As much as two tons of sponge-iron were obtained per day from one furnace. It was principally used in open-hearth steel-making, the loss on conversion into malleable iron in the forge being considerable. The ore treated was a dense, pure, specular hæmatite, containing 66 per cent. of iron.

Siemens' Process.—In 1873 the late Sir C. W. Siemens introduced a direct process of producing malleable iron from the ore, which was afterwards experimentally tried at Birmingham, Sheffield, and Towcester, as also in the United States.

The furnace, of the rotative kind, somewhat similar in character to that of Danks, was provided with four regenerators arranged for gas-firing. The rotative chamber had an aluminous lining, made by grinding a bauxite containing alumina 52·6, ferric oxide 42·26, and silica 4·12 per cent., with 3 per cent. of clay and 6 per cent. of black-lead. This, when very strongly heated, is converted into an infusible substance very similar to emery. The cylinder is closed by a door at one end, and at the other is a connection with the gas-passages and regenerators. The flame on entering is directed to one side of the cylinder and returns by the other, the admission- and discharge-ports being only separated by a thin partition wall. The lining is fettled with hammer- or roll-scale and cinder to a depth of 6 or 8 inches, and when soft, lumps of hæmatite or rich cinder are thrown in to roughen the surface and divide the charge while rotating. The materials used were calcined Northamptonshire ores, containing 40 per cent. of iron and above 2 per cent. of phosphorus, calcined blackband of 60 per cent. produce, and some mill-furnace cinder. Limestone was added sufficient to form a moderately basic slag. Coarsely powdered bituminous gas-coal was added as a reducing material, to the extent of from one-sixth to one-fourth of the total weight of the ore and fluxes. The latter were broken to the size of a pea, and the whole thoroughly mixed.

According to the size of the furnace the charge varied from 24 to 50 cwts. After the gas was turned on, the furnace was left stationary for a few minutes and then slowly turned at the rate of 12 to 15 revolutions per hour. After 2½ or 3 hours the charge began to clot and some fluid slag appeared; the quantity of the latter increased with the temperature, and was partly tapped off from time to time. Finally, the furnace was raised nearly to a white heat, and the reduced iron was balled up by a rabble introduced through the end door, in the same way as in a puddling-furnace. The balls were shingled under a steam-hammer. The time required for one operation was from 4½ to 6 hours; equal to four or five charges per day of twenty-four hours. The yield of blooms was 73 to 77 per cent. of the total iron contained, or from 23 to 27 per cent. passed into the slag.

The consumption of coal was from 32 to 34 cwts. for heating, and 8 to 9 cwts. for reduction on the ton of blooms made, but under the most favourable conditions, with rich ores, it was brought down to 30 cwts. The iron, although made from highly phosphatic ore, was very pure, and contained phosphorus 0·07, sulphur 0·03, and carbon 0·12 per cent.

| The slags contained, | $FeO$ | $Fe_2O_3$ | $Al_2O_3$ | $SiO_2$ | $MnO$ | $CaO$ | $CaS$ | $P_2O_5$ |
|---|---|---|---|---|---|---|---|---|
| From first tapping, | 46·95 | ... | 16·50 | 28·10 | 0·49 | 0·29 | 2·32 | 5·22 |
| From final tapping, | 49·24 | 7·05 | 20·40 | 18·80 | trace | ... | 0·91 | 3·46 |

The amount of iron contained is about the same as in those of the American bloomery process.

This process was not found to be advantageous for making wrought-iron, as the balls were of a loose, spongy texture, and difficult to clean from the slag, which was rather refractory, so that a second heating was required. It was afterwards successfully adopted in making sponge-iron to be used in open-hearth steel-making both at Landore and in the United States.

The direct processes described in the preceding pages may to a great extent be considered as obsolete, or likely to become so in a very short time, as none of the newer modifications have been permanently successful. The older forms have the great advantage of not requiring any large or varied supply of materials, such as fluxes and building materials, the furnaces being built of any sandy clay or mud, and the ore fluxes its own earthy waste. The iron is of excellent quality, even when the ores are somewhat inferior, and better than that made from them by more elaborate processes; but against this must be placed the enormous waste of fuel and labour consequent on the small scale of the operation. There is also a large amount of ore wasted owing to the consumption of iron as a fluxing material, but this is of less consequence where it can be easily obtained.

## INDIRECT METHOD OF OBTAINING IRON—PRODUCTION OF PIG-IRON AND SUBSEQUENT CONVERSION INTO MALLEABLE IRON.

When finely divided iron is exposed to the action of carbon, whether solid or as a gaseous compound, at a high temperature for a sufficient length of time, it becomes carburetted to a maximum, producing cast-iron which is fusible, and which may be collected in a molten state. This condition is always realized to some extent, even in the smallest Hindoo furnaces, where particles of reduced iron, that have been separated from the main mass in the hearth and become entangled in the residual slags, remain for a sufficient time under the influence of the carbonaceous fuel, and are cemented into well-melted globules of cast-iron. These are collected by breaking up and washing the slag, and are used as slugs or buckshot for large game-shooting by the natives, and also for steeling the edges of iron tools. The small bulk of the particles of molten metal in this case keeps them isolated in the viscid slag, but when they are larger and more numerous, they overcome the resistance of the fluid slag and run together, so as to collect at the bottom of the furnace in one mass. This is what actually happened in the working of the Stückofen, when liquid cast-iron was commonly run out in some quantity with the slag before drawing the reduced lump of iron; and it is generally supposed that the modern blast-furnace has been developed from the Stückofen. For the systematic production of cast-iron, however, something more than a mere increase in the size of the furnace is necessary, namely, the substitution of metallic bases (calcium, magnesium, and aluminium), whose oxides are not reducible by carbonic oxide, for iron, as a means of fluxing the silica of the ore and fuel. In other words, earthy fluxes, such as limestone, dolomite, and, to a less degree, shale and bauxite, are used to produce fusible slags, instead of sacrificing a considerable proportion of the iron in the ore, as is done in all direct processes. By appropriately varying the fluxes, the reduction of the ore may be accelerated or retarded, and the simultaneous separation of other elements, such as phosphorus and silicon, and their combination with the reduced and carburized iron promoted or impeded within very considerable limits, so that products differing widely in physical properties may be obtained. All of these, however, have the common property of fusibility at the maximum temperature prevailing in the furnace, and therefore bear the common name of cast-iron.

The result of dry assay shows that cast-iron may be rapidly produced from the ore when it is intimately mixed with reducing carbon and the necessary fluxes, and heated from an exterior source. This, however, is not practicable on a large scale, where the heat, produced by the rapid oxidation of carbon at the bottom of the furnace, is made to act upon the materials or burden in the body of the apparatus by passing the highly heated gaseous products of combustion through them. This supposes that these substances shall be so loosely stacked as to allow a free passage for the gases, while there is practically sufficient surface for the exercise of their reducing action. In other words, the ore and flux must be in fragments of appropriate size, dust ores being excluded as choking up the circulation of the gases, while excessively large lumps may pass through unreduced, producing ferriferous slags and other derangements in the furnace. It is therefore often necessary to subject the materials to a preliminary breaking, and more generally to calcination, and as these operations are of considerable importance, they will be considered before discussing smelting operations proper.

## PREPARATION OF IRON ORES.

WASHING, &c.—The pisolitic ores of Central France, and to some extent those of South Germany, are often associated with a considerable quantity of clay and sand, from which they may be in great part freed by washing in a current of water, by a pug-mill, or by some analogous contrivance; the rough ore, when compact, being first broken by suitable machinery or by hammers. Various simple contrivances of this sort will be found in the older metallurgical works. At the present time the principal apparatus used is the washing-drum or *trommel*. This is a cylinder or cone of wrought-iron 10 to 15 feet long, with its axis inclined about

15 degrees from the horizontal position, which receives a slow motion of rotation by toothed gearing. The stuff is fed in at the upper end and travels slowly down the inclined surface, while a jet of water traverses it in the opposite direction. This carries the sand and lighter waste out at the top, the cleaned ore being delivered below. The ore remains about ten minutes in the drum, or about 10 tons of stuff are washed per hour. A six-horse engine is sufficient to drive it. This includes the power necessary for an elevator lifting the ore about 30 feet.

In order to attain regularity in working, it is desirable that the charges of ore and fluxes should be reduced to fragments of nearly uniform dimensions. The size of these should be regulated in accordance with the height of the furnace and the greater or less degree of facility with which the ore is capable of becoming reduced. In the hæmatite districts of Lancashire, it is usual to break both ore and fluxes to the size of ordinary road-metal; and in Sweden the hard magnetic ores of the country are, after roasting, reduced to a still smaller size; the ore in the Cleveland district is, on the contrary, charged into the furnaces in blocks from 4 to 6 inches cube. These large masses can, however, be only used in furnaces of great height, in which, by the slow descent of the charges, sufficient time is allowed for the complete permeation of heat through them, while their size allows a free passage for the upward current of gases. Smaller pieces, although exposing a larger surface to the action of the reducing gases of the furnace, pack more closely together, and consequently offer greater resistance to the blast. The ores may be broken either by manual labour or by machinery, but there are now but few localities where the older methods of breaking can be advantageously employed, unless hand-picking, for the removal of siliceous gangue or of some injurious associated mineral, is at the same time resorted to. Various contrivances are employed for the reduction of iron ore to fragments of a size suitable for the blast-furnace, but of these the well-known Blake's stone-breaker is most generally used.

A machine of this description, with crank-shaft making from 200 to 250 revolutions per minute, and having jaws 20 inches in depth and from 9 to 10 inches in width, will, with an expenditure of 15-horse power, crush from 10 to 12 tons of hard red hæmatite per hour into fragments containing, on an average, 4 cubic inches.

The corners of the movable jaw are liable to become worn or broken off by use, and the crushing-faces are therefore so made as to admit of being readily replaced in case of accident or when worn out. The most serviceable material for the crushing-surfaces is obtained by the use of a mixture of mottled cast-iron and franklinite spiegeleisen cast in chills. Softer ores require proportionately less power; from 3 to 4-horse power is sufficient for spathic ores and limestone.

At Eisenerz in Styria, crushing-rolls, somewhat similar to those employed in Cornwall for crushing copper ores, are used for the reduction of ironstone to a suitable size for the furnace. These are chiefly made use of for breaking ores which have been previously roasted, and appear to be very efficient for this purpose.

A pair of such rolls, set at a distance of 1½ inch apart and provided with springs to prevent breakage, in case of hard pieces of raw ore being introduced, will, with an expenditure of about 20-horse power, reduce 40 tons per hour of roasted spathic ores to a size suitable for the furnace. In order to do this, the rolls make about 36 revolutions per minute, and the ore, when introduced in fragments of from 20 to 30 cubic inches, is delivered at a maximum size of from 4 to 5 cubic. inches. By slightly reducing the distance between the rolls, and increasing their speed to 42 revolutions per minute, from 60 to 70 tons of calcined ore may be broken per hour with an expenditure of about 24-horse power.

Tilt-hammers and stamps are used for breaking and crushing magnetic ores in Sweden and in the Adirondack district in New York, but these are mostly survivals of antiquated establishments.

WEATHERING OF IRON ORES.—The argillaceous nodular ironstones of the Coal-measures is often contaminated with fragments of adhering shale, which, when first raised, are not readily separated, but by exposure to atmospheric influences oxidation and exfoliation are induced, and the removal of the associated rock becomes easy. Ores also which contain sulphides, such as iron or copper pyrites, when exposed to air and moisture, become gradually purified by the formation of soluble sulphates, which are finally, to a great extent, removed by rain. This process of weathering iron ores with a view to the removal of sulphur in the form of sulphates is much more effective when applied to them after roasting, and this method of treatment is consequently most frequently adopted.

When spathic ores are subjected to weathering in their raw state, the oxidation of sulphides and the removal of the resulting sulphates is usually accelerated, during dry weather, by lixiviating the heaps with water, and also by occasionally turning them over so as to expose fresh surfaces. Hard siliceous ores are, in the Harz, subjected to this treatment for several years before being smelted, but it is evident that this could only be carried on in the case of works of limited capacity, since, for a large establishment, the amount of ground which would be thus occupied would be so great as to render the operation practically impossible.

Spathose ores, when thus treated, experience at the same time a superficial transformation into brown hæmatite, and the same change takes place, to a more limited extent, with nodules of clay ironstone, more particularly when carbonaceous matter is not present, or when its amount is very small.

It is, however, important that care should be taken that the weathering of the ores be not carried too far, as they would then be liable to fall into powder, and become unfitted for treatment in the blast-furnace.

Ores containing much calcium carbonate cannot be subjected to a prolonged weathering after calcination, since the slaking of the lime produced during the process of roasting would lead to the disintegration and crumbling of the ore.

In some of the smaller iron-works in Germany siliceous hæmatites and magnetites containing pyrites are, after crushing and washing, exposed to the air for a period of from two to three years, in heaps of about three feet in height, and are, during that time, repeatedly washed with water. After being thus treated, they are again passed through the crusher, and lixiviated with water during the whole of the following summer. At Altenau, ores of this class are, after roasting, exposed for one year to the action of the rain and air before smelting.

Carbonate of calcium in such ores is, however, a great obstacle to the removal of the sulphur by lixiviation after roasting, since the caustic lime formed decomposes the soluble sulphates of iron and copper, causing those metals to be deposited in the form of hydrated oxides, with formation of calcium sulphate or gypsum. Calcium sulphate, although, to a certain extent, soluble in water, is much less so than the corresponding salts of iron or copper, and, consequently, the sulphur is removed with difficulty; under these circumstances any copper which may be present is retained by the roasted ore.

The removal of phosphoric acid from roasted ores by hydrochloric acid has been attempted, but without success; at Kladno, in Bohemia, sulphurous anhydride was employed for the same purpose, and the results are stated to have been satisfactory; but since the introduction of the Thomas and Gilchrist process such treatment has become unnecessary.

ROASTING OR CALCINATION OF IRON ORES.—All iron ores, with the exception of certain varieties of magnetite and massive red hæmatites, are usually subjected to a process of roasting or calcination before being taken to the smelting-furnace. By this treatment water, carbonic anhydride, and other volatile matters are expelled, and, as the fragments of ore retain very nearly their original size and form, they thereby acquire a degree of porosity which materially facilitates the changes which they subsequently undergo in the furnace. The roasting of iron ores has also the effect of decomposing metallic sulphides, such as iron pyrites, the whole of the sulphur being ultimately expelled, if the temperature be sufficiently high; the metal remaining as an oxide. Ferrous compounds, such as spathic ores, absorb oxygen, chiefly with production of magnetic oxide, but when magnetite is itself subjected to calcination it often becomes externally covered with a coating of peroxide. This action of oxygen, although resulting in a greater expenditure of fuel for the reduction of the iron to the metallic state, has an important practical advantage, since ferric oxide is much more indifferent to the action of silica at high temperatures than is ferrous oxide; the latter, under such conditions, forming with it a highly basic slag, from which metallic iron can only be obtained with difficulty.

The roasting of iron ores is effected either in *clamps* or open heaps, in heaps within walled enclosures, or in furnaces or kilns.

*Roasting in Open Heaps.*—This method is principally resorted to in localities in which fuel, as compared with the price of labour, is cheap, but it has the disadvantage of not only uselessly consuming a large amount of wood or coal, but, from the imperfect distribution of the heat, the interior of a pile is often fused into a compact mass before other portions are sufficiently roasted.

At Königshütte, in the Harz, calcareous iron ores are roasted during from eight to fourteen days in heaps of the shape of a truncated pyramid, 60 feet square at the base, and 9 feet in height.

The floor on which these are built is made of slag, upon which is laid a bed of ironstone 6 inches thick, on which a stratum of small coal of the same thickness is placed. Alternate layers of ironstone and coal are then added, in such a way that towards the top the thickness of the layers of ore is increased to 10 inches, while that of the coal decreases to 3 inches. One cubic foot of small coal is required to roast 3 cubic feet of ironstone.

Blackband ores are generally roasted without any addition of fuel, as they usually contain a sufficient amount of combustible matter to burn by themselves when once fairly lighted. Some varieties of blackband require a small addition of fuel, but in the case of such ores their roasting is much facilitated by removing the smaller fragments by screening.

The blackband ores of Hasslinghausen, in Westphalia, contain from 15 to 30 per cent. of carbon, and are roasted, without addition of fuel, in heaps three feet in height, and of any convenient length and breadth. At Heinrichshütte similar ores are roasted during from one to three months, in heaps 10 to 15 feet in height, 20 to 30 feet in breadth at the base, and of varying lengths. Ores containing a large percentage of carbon are calcined in larger fragments than those which are comparatively poor in combustible matter.

Ironstone very rich in carbon should, in order to prevent caking, be calcined in heaps not exceeding 3 feet in height, whilst ores poor in coaly matter may be roasted in heaps of much greater height. In order to ignite these heaps, they are either surrounded by a channel filled with wood, or holes are left in them at regular distances, which are either filled with wood or with glowing iron ore from a heap already in process of calcination. The regulation of the temperature of open heaps is somewhat difficult to manage, and, consequently, a partial fusion of the ore sometimes takes place, whilst at others the roasting is not complete in certain portions of the pile.

To prevent this irregularity in the results obtained, the following arrangement is adopted in some parts of Westphalia. Heaps 120 feet long, 30 feet broad, and 4 feet high are enclosed between walls built of the larger pieces of ore, small openings being left at intervals of about 12 feet along the sides. These draught-holes communicate with passages in the interior of the heap, filled with wood. The larger blocks are placed along these passages, whilst the finer ore is piled against their sides in order to admit the flame as much as possible into the interior of the heap. After the pile has become fully ignited, the surrounding wall of ore is taken down and thrown upon places in which the fire may exhibit a tendency to come too quickly to the surface. A heap of the above dimensions will contain about 17,000 cubic feet of ore, and usually takes a month to burn completely out.

Ores containing a large amount of coaly matter and pyrites are very liable to become so highly heated that the fragments near the middle of the heap become fused into large masses. To avoid this caking, such ores are, in Westphalia, roasted in heaps only 2 feet in height, and, when this has burned out, but before it has cooled, a similar layer is placed upon the first; in some cases a third is added after the second has burned out.

As the presence of reducing gases tends to prevent the complete oxidation of sulphur, Grundmann recommends that the heaps, when they contain much pyrites, should be covered with a coating of small ore in order to condense the sulphur volatilized without oxidation. This coating is afterwards carefully taken off; the sulphates remaining in the heaps being removed by long-continued exposure to air and occasional watering. He also recommends piling the blocks of ironstone with their planes of stratification in a vertical position, to facilitate the escape of sulphur; since the pyrites contained in such ores is usually found interlaminated between the divisional planes.

In South Wales and Staffordshire the calcination of ironstone in clamps is generally effected as follows:—A bed of coal, a few inches in thickness, is spread upon the level surface of the ground, and covered with a layer of ore about 1 foot in depth; this is followed by fresh layers of coal and ironstone, until the height of the pile has reached from 4 to 5 feet. It is then lighted at bottom, and continues to burn until the whole of the fuel has been consumed. Should the fire in any part of the heap come too rapidly to the surface, it must be damped with some ore or ashes, in order to prevent a partial fusion of the mass; the loss in weight varies from 28 to 33 per cent., and about 2 cwts. of small coal and ½ cwt. of large coal are consumed for each ton of ore roasted.

In Staffordshire and Scotland blackband ironstone is calcined in piles which have generally a trapezoidal form, and vary from 3 to 9 feet in height. In order to avoid the production of too high a temperature the smaller heaps are to be preferred for ores containing a large proportion of combustible matter. The spathic carbonates of South Wales, locally known as "coal brasses," are particularly liable to become fused during the process of calcination, and should consequently be roasted in heaps of very moderate dimensions.

*Roasting between Walls.*—By roasting within walled areas, the draught, and consequently the temperature, can be more easily regulated than in heaps not so protected; a better calcination is also effected with a smaller expenditure of fuel. The expense of both fuel and labour is, however, greater than in kiln-roasting, and the results obtained are less constant and uniform. Three sides of a square, or rectangular area, are usually enclosed within walls, of which the height may vary from 6 to 12 feet, according to the nature of the ores to be calcined, and the floor is in such cases more or less inclined. Two ranges of draught-holes, about 4 inches square, are left in these walls at regular intervals, the lower series being close to the ground and the upper row about 3 feet above them. When the enclosed area is very large it is provided with air-shafts, which are made by building up large masses of ore, so as to form chimneys in the interior of the heap, to which air has access through a system of flues formed on the floor. These flues are sometimes replaced by a layer of wood, so arranged as to admit of a circulation of air to the chimneys. This method of roasting is not employed in this country, but is practised to some extent in the Harz, where clay ironstone is calcined with an expenditure of from 6 to 8 per cent. of charcoal-dust or breeze.

*Roasting in Furnaces or Kilns.*—This method of calcination is generally to be preferred to the ruder processes before described, since it is not only more economical as regards the consumption of fuel, but the temperature is also more completely under control; the product obtained is consequently of a more uniform character. The furnaces or kilns are generally so constructed as to allow of the operation being carried on continuously, the raw ironstone being introduced at the top, while the calcined ore is withdrawn from the bottom. The construction of the kilns employed for this purpose varies considerably in different districts; but the principle of working is, with but few exceptions, everywhere the same. A layer of fuel is first placed at the bottom, and this is covered with layers of ore and fuel alternately, until the internal cavity of the apparatus has been filled to the top. The ore, as it becomes roasted, is withdrawn at the bottom, where air is admitted, and the next layer descends to take its place, the deficiency being made good by fresh charges of ore and fuel at the top. In exceptional cases, the heat is supplied by fuel burned in side-grates, in such a way that the flame and heated gases only have access to the interior of the kiln. In Sweden, instead of using solid fuel, the roasting of iron ores is frequently effected by heat developed by the combustion of waste gases from the blast-furnace.

Kilns of moderate size are usually made either of a cylindrical or conical form, but very large ones have generally either a flattened, elliptical, or rectangular horizontal section; the corners in the latter case being rounded off. In large kilns with a circular horizontal section it is found difficult to maintain a uniform temperature, and the ore in the centre is consequently liable to agglomerate from becoming too highly heated.

At the mines of Gollrath, near Mariazell, Styria, the ores chiefly consist of undecomposed spathic ironstone, with small quantities of brown hæmatite, resulting from the decomposition and oxidation of carbonate of iron. The calcination of these ores takes place in continuously working kilns, in which coal-dust, which could not otherwise be utilized, is the fuel employed. The form and dimensions of the furnaces employed vary in accordance with the nature of the ores to be treated, lump ore being generally roasted in round kilns, 9 feet in diameter and 11 feet in height. The finer ores are, on the contrary, roasted in a large furnace, 80 feet in length and 35 feet in width, of which figs. 38, 39, and 40 represent a longitudinal, horizontal, and transverse section respectively.

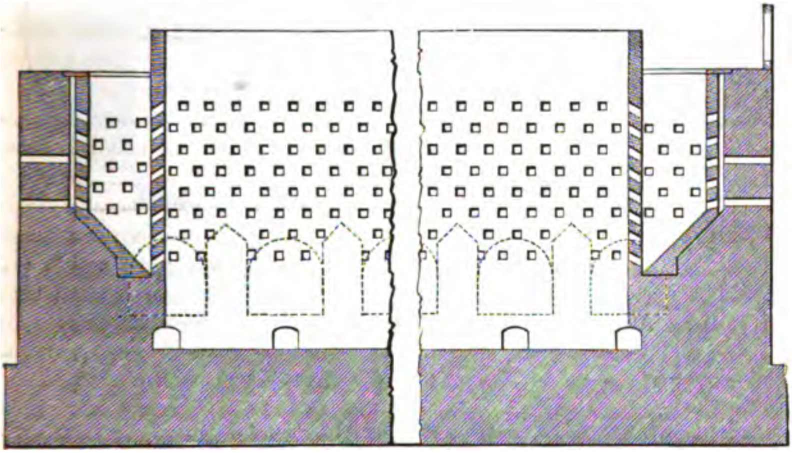

Fig. 38.—Roasting-Kiln, Styria; longitudinal section.

The complicated system of air-holes shown in the drawings is necessary on account of the comparatively fine state of division of the ores, which interferes with the draught, and tends to prevent the regular calcination of the ironstone. This apparatus is charged with alternate layers of coal-dust and ore, in the proportion of 12 to 15 tubs, of 7¾ cubic feet each, of the former, to about 500 cwts. of the latter; these proportions are, however, more or less varied according to the working of the ore. The roasted ironstone is withdrawn from the kilns every six hours, care being taken not to remove any fragments that may not have become thoroughly red-hot. The calcined ore thus raked from the bottom of the kilns is slaked on the drawing tables, *a*, by means of water conveyed through pipes for that purpose, and issuing from perforated roses, *b*. The excess of water employed, passing through gratings, *c*, escapes by the gutters, *d*.

After being thus slaked, the ore is weathered during from four to six years, and is periodically watered in order to remove the soluble salts

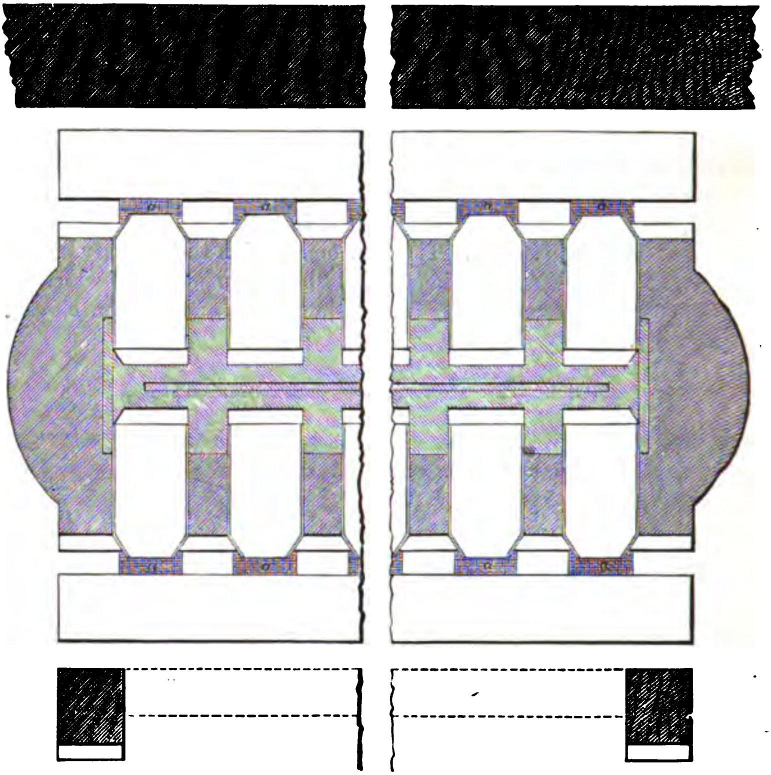

Fig. 39.—Roasting-Kiln, Styria; horizontal section.

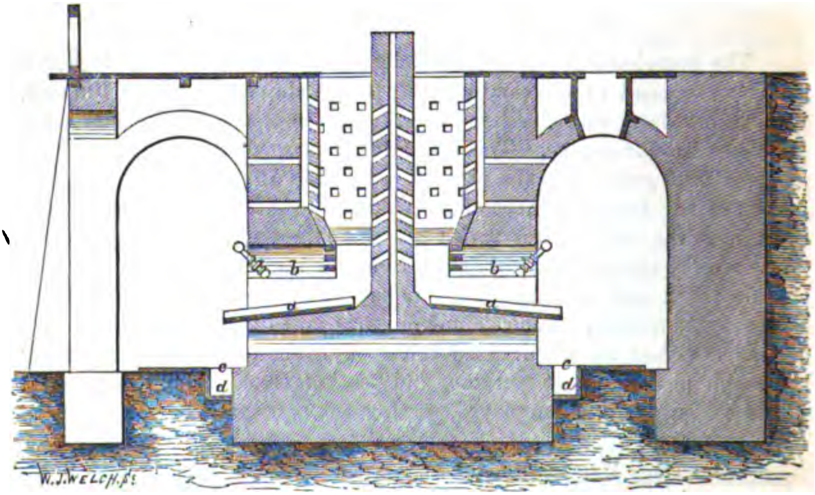

Fig. 40.—Roasting-Kiln, Styria; transverse section.

formed. The average expenditure of fuel is at the rate of 0·23 of a cubic foot of coal-dust per cwt. of ore roasted, and the loss of weight experienced amounts to about 22 per cent.

At Altenberg six continuously working circular kilns, each 9 feet in internal diameter, and having a total depth of 10 feet 5 inches, are found sufficient to supply the three blast-furnaces at Neuberg. These kilns are provided with step-grates, as shown in fig. 41, which represents a vertical section of one of them by a plane passing through the centre of the grate. The masonry, *a*, is of ordinary rubble-work, but the interior of the kiln is provided with a refractory lining, *b*; the grate, *c*, is of cast-iron, and the bottom, *d*, is composed of plates of the same material.

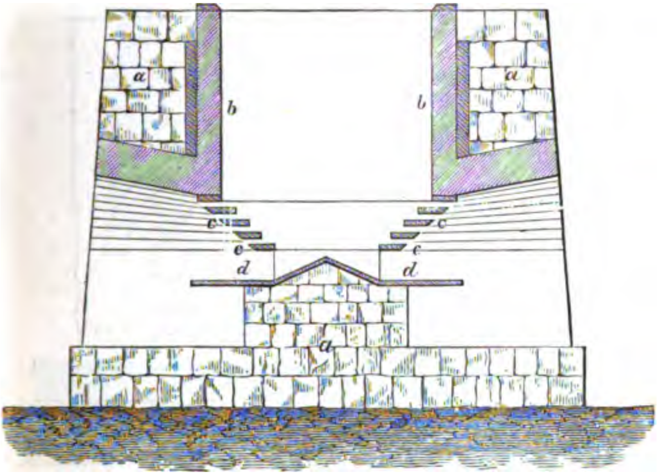

Fig. 41.—Roasting-Kiln, Altenberg; vertical section.

The ores roasted contain from 5 to 6 per cent. of magnesia, and are exposed for several years to the action of the atmosphere without lixiviation. From 23 to 24 per cent. of their weight is lost in roasting; about 0·35 of a cubic foot of charcoal-dust is employed per cwt. of ore roasted.

At the Dowlais Iron-Works, in South Wales, the kilns used have a rectangular form with rounded ends; the width at top being 9 feet, and at bottom 2 feet; their length is 20 feet, and their height 18 feet. Fig. 42 is a front elevation, fig. 43 a longitudinal section, and fig. 44 a transverse section of one of these kilns, as given by Truran. The floor consists of cast-iron plates 2 inches in thickness, and the interior is lined with fire-bricks; in front are two splayed arches extending backwards to the refractory lining of the kiln, and in each of these are two rectangular openings, *a*, on the floor-level, through which the calcined ore is withdrawn previous to being filled into barrows or waggons for conveyance to the furnace. Above these openings is a series of apertures, *b*, employed for the purpose of regulating the draught. The top edge of the kiln is covered by a flanged cast-iron ring, *c*, which protects it from abrasion during the process of filling. The operation of calcining in these kilns is conducted in the following way:—Two or three coal fires having been lit on the floor, raw ironstone is placed on top and around them, until the whole floor is covered with ironstone at a dull red heat; a fresh layer of ironstone, about 9 inches in thickness, is then added, mixed with about 5 per cent. in weight of small coal, and as soon as this stratum has become heated to

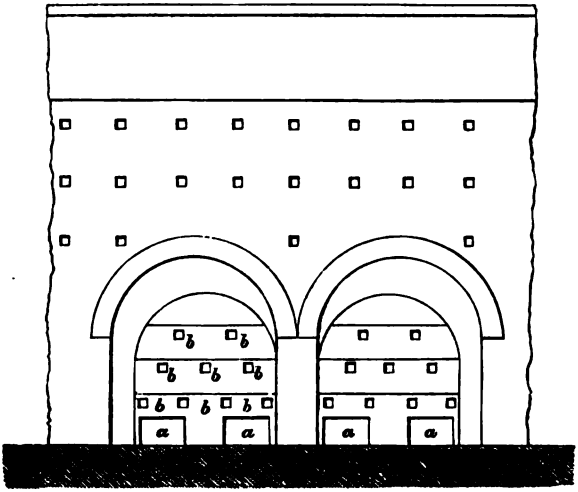

Fig. 42.—Roasting-Kiln, Dowlais; front elevation.

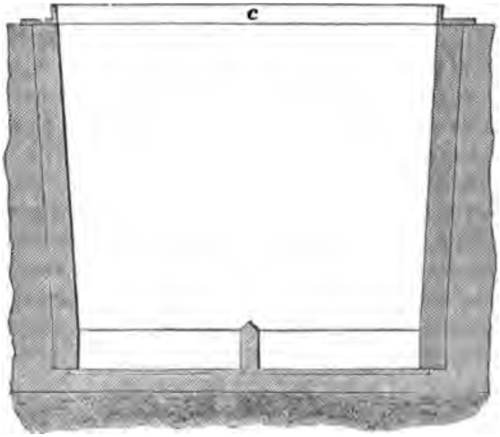

Fig. 43.—Roasting-Kiln, Dowlais; longitudinal section.

redness another is added. This addition of fresh layers of raw ironstone and small coal is repeated as fast as the previous layers become heated to the requisite temperature. In this way the kiln is filled to the top, and by the time this has been done the lower portions, which were first ignited, will have become sufficiently calcined for drawing; fresh charges are thus added at top, so as to occupy the space left by the sinking of the mass on the inside, caused by the daily withdrawal of calcined ore from the bottom.

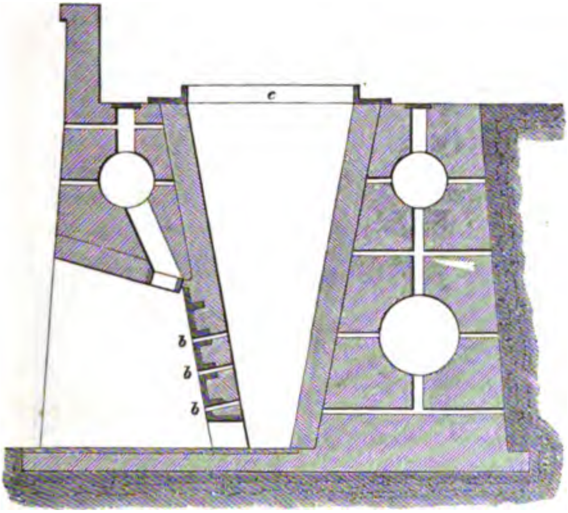

Fig. 44.—Roasting-Kiln, Dowlais ; transverse section.

The capacity of a kiln of the dimensions given is about 70 tons, and the amount of ore calcined weekly amounts to 146 tons; the consumption of small coal is at the rate of 1 cwt. per ton of ore calcined, whereas, in clamps, 2 cwts. of small and ½ cwt. of large coal are required to do the same amount of work.

Welsh argillaceous ores generally lose 27 per cent. of their weight by calcination; blackband ironstone from 40 to 60 per cent.; red hæmatite about 6 per cent.; and Cornish, Devonshire, and similar brown hæmatites from 12 to 14 per cent.

In the Cleveland district Gjers' calcining-kilns are extensively employed. Fig. 45 is in part an elevation and in part a section of this kiln, which consists of a body or shell of fire-bricks only 14 inches in thickness, cased externally with wrought-iron plates. The interior diameter, at top, is, in the older kilns, 18 feet; at the boshes, or widest part, forming the junction of the two truncated cones, 20 feet; and at the bottom 14 feet; the horizontal section is everywhere circular. The bottom of the brickwork rests on a cast-iron ring, *a*, 4 inches in thickness, which is supported by cast-iron pillars, *b*, each 27 inches in height; thus leaving an open space between the bottom of the kiln and the floor. The floor is covered by iron plates *c*, 2½ inches in thickness, cast in segments, and carrying, in the centre, the cone, *d*, 8 feet in diameter at the base, and 8 feet in height. The total depth from the filling-gallery, *e*, to the foundation-plate, *c*, is 24 feet, and its internal capacity is 5,500 cubic feet. As in the kilns of South Wales, the amount of fuel required amounts to about 5 per cent. of the weight of ore calcined. The roasted ore is drawn through the openings between the pillars, and is directed outwards by the slope of the central cone. The height of the newer kilns

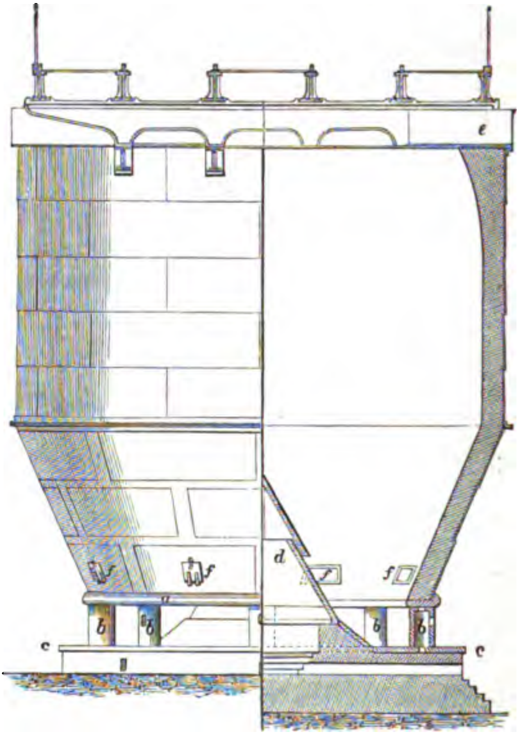

Fig. 45.—Gjers' Calcining-Kiln; one-half in section.

of this description is 33 feet, the diameter 24 feet, and the cubic capacity 8,000 feet. Such a kiln is capable of calcining 800 tons of iron ore per week, and will burn 24 to 25 tons with one ton of small coal. Around the lower tier of plates are a number of openings, *f*, ordinarily closed with doors, which are occasionally useful in case of the stone becoming agglomerated. A double roadway passes over all the kilns, with a gangway between and outside the two roads.

*Gas Calcining-Kilns.*—The use of the waste gases of the blast-furnace, instead of solid fuel, for calcining iron ores, is restricted to a very few localities in Styria and Sweden. The Styrian kilns are of two kinds. Those on Fillafer's principle, intended for ore in lumps, have upright rectangular stacks, each 8¾ feet high, 4⅔ feet long, and 1¾ feet broad, with a capacity of 70 cubic feet. The stack is covered with a dome and chimney for carrying off the volatile products. The ore is charged through a door near the top, and drawn at the bottom by an inclined plane faced with cast-iron plates. The gas is introduced through a number of rectangular slits in the wall of the stack, where it mixes with air introduced through other apertures of the same kind. Twelve kilns are required for one blast-furnace. They are built in series of six back to back, and are often at such a height that their tops are level with that of the furnace. About 40 tons are calcined by such a set of kilns in twenty-four hours.

Moser's kiln for calcining small and wet spathic ores has a bed from 4 to 5 feet wide, and from 15 to 18 feet long, under a low arched roof inclined about 20° from the horizontal. This is filled with flame from a series of gas-jets in the roof at the lower end, while the ore is introduced at the upper end of the bed, whose slope is just sufficient to allow the charge to rest, except when calcined ore is drawn out at the bottom. This kiln was originally intended for drying the ore only, but has been found to produce perfect calcination when properly regulated. About 15 tons of ore may be dried, or 7½ tons calcined, in twenty-four hours.

Westman's gas-kiln, which is used extensively in Sweden for calcining magnetic ores, has a circular stack, somewhat like the upper part of a blast-furnace, built of fire-brick, with an outer casing of red brick hooped with wrought-iron. The largest size is about 18 feet high, 6 feet in diameter above, and 10 feet below. The base of the kiln is a low pyramid of from 5 to 8 faces, which leads the calcined ore outwards to the drawing-arches, which are faced with cast-iron plates and closed with iron doors. The gas is introduced from a ring main encircling the lower part of the stack, through a series of passages or burners of which the number is double that of the drawing-passages, air being supplied through parallel passages above. In addition to these there are two rows of stoking-holes in the stack for introducing bars in case of the charges hanging in the furnace. The top of the kiln is closed with iron plates having a central counterpoised trap-door for charging. The volatile products are carried off by four parallel sheet-iron chimneys about 12 feet high. The descent of the charge is regulated by the removal of the calcined ore, which is drawn at short intervals from each passage in succession. Unlike the ordinary practice of working, a strong clotting heat is used in order to decompose any iron pyrites included in the ore. The estimated consumption of furnace gas is about 10,000 cubic feet, corresponding to 160 or 180 lbs. of coal per ton of ore. About 5 tons of ore are drawn from each passage in twenty-four hours; it is at a strong red heat, and must be removed in iron waggons. In some instances air under pressure from the blowing-engine of the furnace has been used in these kilns as well as steam, to promote desulphurizing, but never with very marked success.

Calcining-kilns may also be heated by gas prepared in special producers, but this plan is obviously disadvantageous so far as economy of heat is concerned, and is only valuable as a means of keeping the ore free from the ash of the fuel. On the latter account it is sometimes used in the construction of limekilns, where a very pure product is required.

The calcination of iron ores is now almost entirely restricted to black-band, spathic, and other carbonates. Limonite and other hydrated ores, although they contain a large proportion of water, are generally smelted as such, since many of them fall to powder when dehydrated, while such of them as contain sandstone and quartz are likely to form fusible silicates in the kiln, and under either condition their value as a material for the smelter is considerably lessened.

BLAST-FURNACE SLAGS.—The essential point of difference between the making of cast-iron and of wrought-iron at one operation from the ore, is the substitution of oxides of some one or more metals other than iron for the removal of the siliceous or earthy contents of the charge or burden of the furnace. These waste matters are for the most part contained in the ore, but, in a less degree, they are also present in the ash of the fuel. For this purpose it is necessary to use some oxide which is not reducible by carbonic oxide alone, and the only oxides practically available that satisfy this condition are those of manganese, aluminium, magnesium, and calcium; the latter, in the form of caustic lime or carbonate of lime, being of principal importance. The required oxide may be obtained by a judicious mixture of siliceous with calcareous ores, but in by far the greater number of instances limestone is added under the name of *flux* or *fluxing stone.*

The value of limestone for fluxing purposes depends almost entirely on its freedom from foreign matters, especially silica, refractory silicates, and metallic sulphides, as its chief functions are the removal of silica and sulphur from the ores. Theoretically, pure carbonate of calcium contains 56 per cent. of lime, so that for every 100 parts of lime required at least 179 parts of limestone must be used. This fluxing power is very rapidly deteriorated by even a small amount of silica. Thus, supposing 35 parts of silica to require 100 of limestone, considered as pure carbonate, the equivalent quantity would be increased to 121 parts, supposing the stone to contain only 5 per cent. of silica.

Magnesia when present in limestone in the form of dolomite is of value in special cases for the manufacture of forge and manganiferous pig, but is objectionable for smelting foundry-iron. The use of dolomite is therefore generally avoided, even when the works are favourably situated for obtaining it, and pure limestone has to be brought from a distance.

In the United Kingdom good fluxing limestones are common in many geological formations. Among these the most important are the Silurian limestones of Dudley used in South Staffordshire, the Carboniferous limestones of Northern and Central England, and North and South Wales, which not only supply the furnaces in their immediate neighbourhood, but are largely exported to other districts, such as Cleveland and Staffordshire.

Shelly oolitic limestones are used in the furnaces of Northamptonshire and the adjacent counties, and chalk in those of Wiltshire; Devonian limestone in the Liége district, and crystalline limestone from the older formations in Sweden and the United States. In strong contrast to the latter is the use as flux at Newark, New Jersey, of oyster-shells, which are collected from the New York eating-houses.

Another important function of limestone flux is the removal of sulphur, contained in the ore or fuel, as sulphide of calcium, which passes into the slag. The same property is possessed by oxide of manganese, which forms an irreducible sulphide of manganese in the slag. Magnesia, on the other hand, has no power of absorbing sulphur.

In the less frequent condition of ores that contain an excess of earthy bases, such as the spathic ores of Styria and the oolitic ores of Lincolnshire, it is necessary, when siliceous mixing ores cannot be got, to add silica and alumina in the form of clay-slate or shale. The average composition of the latter is:—

| $SiO_2$. | $Al_2O_3$. | $Fe_2O_3, FeO$. | $CaO, MgO$. | Alkalies. | Volatile. |
|---|---|---|---|---|---|
| 60 | 19 | 8 | 3 | 2 | 8 |

Alumina alone may be supplied in the form of bauxite or aluminous iron ore. The most advantageous manner of supplying silica to calcareous ores is as ferrous silicate in the form of forge-cinder, the iron being recovered simultaneously. This is now done in Styria instead of using schist fluxes, as the cinder obtainable is very pure.

The use of caustic lime instead of limestone has been proposed and tried several times, but the saving of fuel which was expected by relieving the furnace from the duty of decomposing the carbonate has not been realized. This is partly due to the circumstance that, under particular conditions of temperature and pressure, carbonic acid may to some extent be re-absorbed by the caustic lime in the upper regions of the furnace, and the carbonate so formed has to be decomposed with a fresh expenditure of heat before actual fluxing commences.

The use of fluxes in the blast-furnace is regulated by several conditions. Where the ores are of high quality and charcoal is used as fuel, the additions may be so proportioned as to give a minimum of slag, and that of the most readily fusible composition, so as to economize fuel. This, however, can only be realized in making white forge-iron with a minimum of silicon and graphitic carbon, and is usually accompanied with the scorification of some of the metallic contents (iron and manganese) in the ore. With the same fuel, therefore, when grey-iron is required, the slag must not only be made more refractory, but its volume must be increased above that required by the condition of minimum fluxing, in order that the reduced iron may have time to take up the necessary metalloids before it is finally melted and collected in the hearth. With mineral fuel, for ores of equal yield, the amount of slag is further increased by the necessity of fluxing a larger quantity of ash, 8 to 15 per cent. as against 2 to 3 per cent. in charcoal, as well as by that of preventing the sulphur (about 0·5 to 0·8 per cent.) from combining with the reduced iron, for which purpose manganous oxide or lime, preferably the latter, must be present, in sufficient quantity above that required for fluxing silica, to form MnS or CaS. A further condition, however, is that these sulphides must not amount to more than a small proportion (3 to 4 per cent.) of the total weight of the slags, and they must also be kept as free from iron as possible. The rule requiring the slags to be more refractory for grey- than for white-iron applies as well to smelting with coke as with charcoal. From the above considerations it may be readily understood that charcoal-furnace slags are more siliceous than those of coke furnaces for metal of the same quality, and that the former may be rendered more refractory by augmenting the proportion of silica, and the latter that of the bases.

The silicates constituting blast-furnace slags may be regarded as ranging between the limits indicated in the following formulæ :—

$2RO.3SiO_2$ or bisilicates with about 60 per cent. $SiO_2$.
$10RO.3SiO_2$ or six-tenth silicates ,, 30 ,,

The nature of the bases is not considered; RO including both protoxide and sesquioxide bases.

The greater number of coke-furnace slags approximate to unisilicates ($2RO.SiO_2$), with the oxygen coefficient ranging from about 0·85 up to 1·25. The corresponding variations in the principal constituents are—

$SiO_2$ 35-40, $Al_2O_3$ 10–20, CaO 30–40, MgO 0–7·5 per cent.

Charcoal-furnace slags are more siliceous, and approximate mainly to bisilicates ($RO.SiO_2$); their oxygen coefficients ranging between 1·25 to 2·25; exceptionally they go as high as trisilicates with the oxygen coefficient 2·75 to 3·25.

The heat required for fusing blast-furnace slags has been elaborately investigated by Åkerman, who has determined, by means of a calorimeter, the actual amount of heat retained at the moment of solidification, both in ordinary slags and in mixtures of silicates of known composition. The average thermal requirement for fusion in 74 slags was 388 units; the extremes being 340 units for the most fusible, which was produced from spathic ore at Vordernberg in Styria, and 463 for the most refractory, a very magnesian slag smelted from titaniferous ore at Lindefors in Sweden. The complete analysis of these slags is as follows :—

| | $SiO_2$. | $TiO_2$. | $Al_2O_3$. | MgO. | MnO. | FeO. | CaO. | Oxygen. Coefficient |
|---|---|---|---|---|---|---|---|---|
| Lindefors . . | 39·20 | 8·50 | 9·59 | 27·25 | 0·53 | 1·54 | 12·65 | 1·08 |
| Vordernberg . | 42·90 | ... | 9·97 | 12·00 | 7·32 | 0·77 | 26·35 | 1·21 |

The melting-points of slags are supposed to range between about 1,100° or 1,200° C., for the most fusible, to 1,500° C., for the most refractory; but these figures are only approximations deduced from calculations, and depend upon assumed values for the specific heat.

The following examples of mixtures intended to produce slags of definite properties are taken from De Vathaire's work;[1] they refer to furnaces burning coke or coal:—

FOR VERY GRAPHITIC IRON.—NO. 1 FOUNDRY OR BESSEMER.

| | 1. | 2. | 3. | 4. | 5. | 6. | 7. | 8. | 9. | 10. |
|---|---|---|---|---|---|---|---|---|---|---|
| $SiO_2$ | 39 | 38 | 37 | 36 | 36 | 36 | 36 | 37 | 38 | 38 |
| $Al_2O_3$ | 10 | 12 | 14 | 16 | 17 | 19 | 20 | 21 | 22 | 23 |
| CaO | 51 | 50 | 49 | 48 | 47 | 44 | 45 | 42 | 40 | 39 |

Nos. 1–3 are ultra-calcareous, 4–7 ultra-basic, and 8–10 ultra-aluminous. The latter will give the most siliceous iron, and are best suited for pure ores or such as contain fusible earthy matters.

| | 11. | 12. | 13. | 14. | 15. | 16. | 17. | 18. | 19. | 20. |
|---|---|---|---|---|---|---|---|---|---|---|
| $SiO_2$ | 38 | 37 | 36 | 35 | 34 | 34 | 35 | 36 | 36 | 37 |
| $Al_2O_3$ | 10 | 12 | 14 | 16 | 18 | 19 | 20 | 21 | 22 | 24 |
| CaO | 42 | 41 | 40 | 39 | 38 | 37 | 35 | 33 | 31 | 29 |
| MgO | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 |

Nos. 11–20 are similar to the preceding series, ten units of magnesia being substituted for an equal amount of lime. This gives greater fluidity, while allowing the amount of silica to be reduced to a minimum.

FOR NO. 3 FOUNDRY-IRON.

| | 21. | 22. | 23. | 24. | 25. | 26. | 27. | 28. |
|---|---|---|---|---|---|---|---|---|
| $SiO_2$ | 40 | 39 | 38 | 38 | 38 | 39 | 39 | 40 |
| $Al_2O_3$ | 11 | 13 | 16 | 18 | 19 | 20 | 21 | 22 |
| CaO | 48 | 47 | 45 | 43 | 42 | 41 | 35 | 37 |
| (Fe,Mn)O | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |

These being intended for a heavier burden than the preceding mixtures, a certain amount of metallic oxide may be unreduced, which promotes fusibility. One unit of iron or manganese allows the burden to be raised 6 or 8 per cent. with the production of mottled iron. No. 21 gives the maximum amount of lime permissible and No. 28 that of alumina.

FUSIBLE MIXTURE FOR WHITE-IRON.

| | 29. | 30. | 31. | 32. | 33. | 34. | 35. | 36. | 37. |
|---|---|---|---|---|---|---|---|---|---|
| $SiO_2$ | 40 | 39 | 39 | 38 | 37 | 38 | 39 | 40 | 41 |
| $Al_2O_3$ | 8 | 10 | 12 | 13 | 14 | 15 | 16 | 17 | 18 |
| (Ca,Mg)O | 48 | 47 | 45 | 45 | 45 | 43 | 41 | 39 | 37 |
| (Fe,Mn)O | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 |

The mixtures in the first part of the above table give better carburized and desulphurized irons than the more aluminous ores at the end. The tendency of alumina to promote reduction of silica may be corrected by magnesia, or preferably by oxide of manganese.

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[1] 'Construction et Conduite des Hauts Fourneaux,' 1885.

FOR MANGANIFEROUS IRONS (SPIEGEL AND FERRO-MANGANESE).

| | 38. | 39. | 40. | 41. | 42. | 43. | 44. | 45. |
|---|---|---|---|---|---|---|---|---|
| $SiO_2$ | 35 | 36 | 34 | 33 | 31 | 29 | 30 | 31 |
| $Al_2O_3$ | 10 | 8 | 12 | 13 | 8 | 12 | 14 | 12 |
| CaO | 45 | 38 | 36 | 36 | 37 | 35 | 33 | 42 |
| MgO | ... | 10 | 10 | 10 | 8 | 8 | 8 | ... |
| MnO | 10 | 8 | 8 | 8 | 15 | 15 | 15 | 15 |

Nos. 38–41 are for spiegeleisen, and Nos. 42–45 for ferro-manganese. Magnesia is used to diminish the proportion of silica and preserve the fusibility, and to save manganese oxide. The amount of the latter base in the slag should not exceed 8 per cent. for high (20 per cent.) spiegel; with 70 per cent. ferro-manganese it may be about 15 per cent., and with 80 per cent. as much as 20 per cent. In the latter case the coke burnt is 50 cwts. per ton of metal.

The proportion of sulphur in slags should not exceed about 2 per cent., corresponding to about 3·5 per cent. of sulphide of calcium. According to Bell, about 94 per cent. of the total sulphur contents of the burden and fuel pass into the slag, and the remaining 6 per cent. into the metal. Manganese oxide is also a powerful desulphurizer. This is well seen in the manufacture of basic Bessemer or Thomas metal at Ilsede, in Hanover, where the furnaces are burdened to produce a white-iron with a minimum of silicon; but owing to the large amount of manganese in the ore it is practically free from sulphur.

The slags produced on the large scale differ from those given in the mixtures by the addition of the alkalies in the charge, which are not usually taken into account in proportioning the ingredients. The following are a few typical examples :—

ANALYSES OF BLAST-FURNACE SLAGS.

| | 1. | 2. | 3. | 4. | 5. | 6. | 7. | 8. | 9. | 10. |
|---|---|---|---|---|---|---|---|---|---|---|
| $SiO_2$ . | 54·26 | 35·80 | 53·79 | 46·37 | 40·25 | 38·48 | 43·07 | 39·52 | 38·76 | 27·68 |
| $Al_2O_3$ . | 6·76 | 4·20 | 13·04 | 4·30 | 8·40 | 15·13 | 14·85 | 15·11 | 14·48 | 22·28 |
| CaO . | 24·56 | 2·14 | 25·67 | 38·64 | 15·50 | 32·82 | 28·92 | 32·52 | 35·68 | 40·12 |
| MgO . | 0·28 | 4·06 | 0·57 | 7·40 | 13·00 | 7·44 | 5·87 | 3·49 | 6·84 | 7·27 |
| FeO . | 9·20 | 21·16 | 2·44 | 0·95 | 10·40 | 0·76 | 2·53 | 2·02 | 1·18 | 0·80 |
| MnO . | 4·80 | 29·64 | 2·20 | 1·86 | 11·67 | 1·62 | 1·37 | 2·89 | 0·23 | 0·20 |
| CaS . | ... | ... | ... | 0·03[1] | 0·07 | 2·22 | 1·90 | 2·15 | 0·98 | 2·00 |
| $P_2O_5$ . | ... | ... | ... | trace | ... | 0·15 | ... | ... | ... | ... |
| Alkalies | ... | ... | ... | 0·45 | ... | 1·92 | 1·84 | 1·06[2] | 1·11[2] | ... |
| | 99·86 | 97·00 | 97·71 | 100·00 | 99·29 | 100·54 | 100·35 | 98·76 | 99·26 | 100·35 |

Nos. 1, 2, 3, 4, and 5, from furnaces fed with charcoal. Nos. 1 and 2. From Rothau; Klasek. 3. From Rübeland; green and glassy; furnace producing mottled iron; Rammelsberg. 4. From Edsken, Sweden;

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[1] Sulphur.

[2] Potash.

producing Bessemer pig; Ullgren. 5. From Eisenerz, Styria; furnace heavily burdened for white-iron. 6. From Dowlais; produced when making grey-iron; Riley. 7. From Dowlais; produced with white-iron; Riley. 8. From cold-blast furnace, working with coke, at Wednesbury Oak, Tipton, South Staffordshire; D. Forbes. 9. Hot-blast furnace, working with coke, near Dudley; Percy. 10. Clarence, Durham; produced from Cleveland ores; Bell.

In general the slags produced in furnaces working with a light burden are either white or grey. This is caused by the almost complete reduction and removal of the iron; but when the ores contain manganese an amethystine tint may frequently be observed. This is particularly the case in charcoal furnaces smelting non-aluminous ores.

A heavy burden and a comparatively reduced temperature, on the other hand, commonly give rise to black or very dark-coloured slags.

The fluidity and vitreous character of slags, are, within certain limits, greater in proportion to the amount of silica present. Opalescent slags usually indicate the presence of a considerable amount of alumina.

Slags containing sulphides of calcium, barium, or manganese, such as are produced when sulphur is present in the ore or fuel, give off sulphuretted hydrogen if allowed to flow over damp ground. If the slag be sufficiently hot to cause the ignition of this gas, it burns with the formation of water and sulphurous anhydride; if, however, on the contrary, sulphuretted hydrogen should escape without decomposition, its presence will at once be recognised by its odour.

If a large quantity of lime be present, the slag has usually a dull, stony fracture, and when the amount becomes excessive, the cinder readily falls to pieces on exposure to a damp atmosphere. If ground with one-fourth their weight of caustic lime, such slags form an excellent cement or mortar for building purposes.

When a furnace is working with a light burden, the slags flow continuously and steadily, and exhibit a somewhat viscid fluidity, passing slowly from the liquid to the solid state. The scouring slags from a furnace working with a heavy burden, on the other hand, flow as freely as water, but become readily solidified without passing through an intermediate state of plasticity. Hæmatites containing manganese produce slags exhibiting the usual amethystine tints characteristic of the presence of that metal; but when blown into bubbles by escaping gases, this colour disappears, and they assume a pearly-white lustre, and pumice-like structure. When furnaces treating such ores are burdened so heavily as to afford white-iron, the slags become dark-green in colour.

In addition to the green and black colours resulting from the presence of ferrous oxide, the amethystine tints produced by oxide of manganese and the yellow and brownish-green colourations due to sulphide of manganese, certain others, particularly shades of blue, are common in blast-furnace slags. Among these a light sky-blue tint, frequently seen in Swedish slags, has been variously attributed to the presence of vanadium and titanium, but the nature of the colouring agent in such cases does not appear to have been satisfactorily determined. Very basic aluminous slags are sometimes speckled with blue from intermixed spinel crystals, which though very minute are of a deep sapphire blue colour.

Various crystallized products have been found in blast-furnace slags, many of which are known as minerals, while others have not as yet been found in nature. Among these are various silicates of magnesium and calcium, such as enstatite ($MgSiO_3$), wollastonite ($CaSiO_3$), and augite $(Ca,Mg)SiO_3$. When the magnesia is less than one-third of the mixed bases in the latter formula, a hexagonal lime silicate, not known as a mineral, is found. In the unisilicate slags, low in alumina, substances of the composition of olivine ($R_2SiO_4$) are common, the bases being lime, magnesia, and manganese oxide, the first predominating; but when alumina forms a suitable proportion, an unnamed tetragonal silicate, somewhat similar to idocrase, is produced. The most characteristic mineral of English slags which are more basic than unisilicates is melilite or humboldtite $12(R.Mg)O.2Al_2O_3.9SiO_2$. This occurs in square tabular or columnar crystals, often of considerable size, in almost every iron-making district, the best examples being found in the slags from cold-blast furnaces. Another tetragonal silicate, which is the most basic of all, is gehlenite ($3RO.Al_2O_3.2SiO_2$), found, but more rarely, in slags produced with a burden containing alumina and magnesia in notable quantity. Under the latter conditions a portion of these bases combine, forming Spinel, which is found in crystals diffused through those of melilite or gehlenite, and, in extreme cases, about one-half of the alumina in the charge may be so combined and take no part in fluxing silica.

Sulphides of calcium and manganese are found mainly as protosulphides diffused through the mass of the crystallized silicates. It is not quite certain, however, whether a higher sulphide of calcium may not sometimes be present.

*Disposal and Utilization of Slags.*—One of the most important points in blast-furnace economy is getting rid of the enormous quantity of slag produced, for which, as a rule, no useful application can be found. To some extent it is disposed of as railway ballast, and in countries deficient in stone, for road metal. In Belgium and Germany it is sometimes slowly cooled in large masses, so as to promote the formation of crystals, with the result of producing a porphyritic substance which can be dressed up into paving setts. In Styria slags are often granulated by allowing a jet of water to impinge upon the molten stream as it issues from the furnace. This produces a voluminous spongy sand, which is used for mixing with mortar, and lies conveniently in valleys for removal by the spring freshets from the mountains. In Cleveland and elsewhere the so-called silicate cotton or slag-wool is made by allowing steam to blow through the liquid cinder, which draws it out like spun glass. This has been used to some extent as a non-conducting covering for boilers and steam-pipes, but the manufacture is not increasing. Granulated slag may also be made into excellent concrete and bricks with Portland cement, or with proper additions of lime, it may even be used as the basis of the cement. For all uses, however, the amount consumed is trifling, and therefore the bulk of the cinder produced has to be tipped to waste, the usual method adopted being to mould it into rectangular blocks, of from 2 to 5 tons weight, upon a cinder-waggon railway-truck, placed alongside the furnace, which, when filled, is drawn away by a locomotive, and, when solidified, thrown upon the waste heap. In America the custom of pouring the liquid cinder out upon the ground is now very often adopted. At the Edgar-Thompson Works, near Pittsburg, the cinder is received in brick-lined tank-waggons, with a valve in the side, holding 10 tons. These, when filled, are drawn to a waste heap about half a mile from the furnaces; the valve is opened, and the liquid cinder runs out on the ground. By this arrangement less land is required for the cinder than when it is moulded into blocks before tipping.

### The Blast-Furnace and its Accessories.

Blast-Furnace.—The blast-furnace, of which an early form is represented in fig. 46 in vertical, and in fig. 47 in horizontal section through the hearth, consists of a shaft or cavity formed of two truncated cones joined together at their bases. The upper and deeper of these cones is known as the *stack* or *body*. The opening at the top of the furnace is called the *throat*, and is often surmounted by a chimney, in which there are one or more openings for the convenience of charging. The lower cone is known by the name of the *boshes*. The lowest division or *hearth*

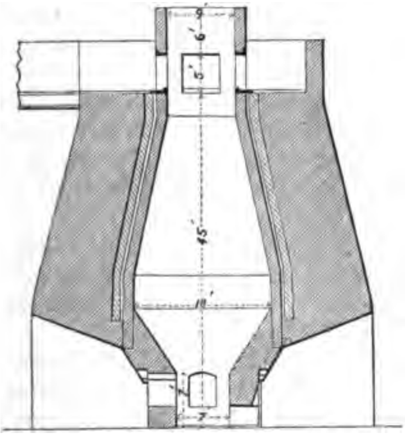

Fig. 46.—Blast-Furnace, Plymouth Iron-Works; vertical section.

is the straight-sided narrow portion below the bottom of the boshes. The broadest part, at the junction of the stack with the boshes, is known as the *belly* of the furnace.

The hearth in the older forms of furnace is supported on a mass of masonry, in which channels are left open for the escape of any moisture which may be expelled from the brickwork; while, to keep the whole building perfectly dry, the foundations are traversed by arched galleries, which intersect each other at right angles beneath the axis of the internal cavity of the furnace. In new furnaces these elaborate substructures are generally abandoned, and the hearth bottom rests upon a bed of concrete, sunk in the ground.

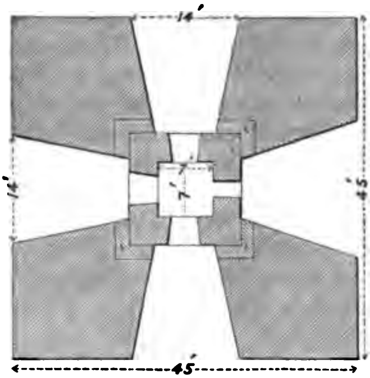

Fig. 47.—Blast-Furnace, Plymouth Iron-Works; horizontal section.

Three only of the sides of the hearth are continued to the bottom the fourth being merely brought to within a certain distance of the base, where it is supported by strong bearers of cast-iron, firmly fixed into the masonry of the walls, and on which rests a block of refractory material called the *tymp*.

At a short distance beneath the tymp, and a little in advance of it, is placed the *dam-stone*, which has a prismatic form, and is securely fixed by a strong piece of cast-iron, which covers its outer side, and is known by the name of the *dam-plate*.

At a short distance above the ground-level, passages for the introduction of the blast are perforated through the walls of the hearth; these are known as the *tuyer-holes*, and usually vary in number from two to six.

The stack and boshes are circular in section throughout. The hearth, as seen in fig. 47, is square, but this construction is now entirely abandoned in favour of the round form.

*Materials and Construction.*—The different parts of a blast-furnace are of very unequal durability; the hearth and boshes being subjected to a much more intense heat than the stack, are sooner worn out, and it is therefore necessary to build the furnace so that these lower portions can be replaced without destroying the upper part. This is done by supporting the stack upon pillars of masonry connected by arches, as in figs. 46, 47, or, what is now more general, upon cast- or wrought-iron pillars or standards, so that the hearth may be freely approached all round. In such cases, the stack rests upon a heavy cast-iron ring, carried on the top of the pillars. The hearth may be built of fire-brick, or of any natural refractory material. Although the former is now more common, the latter is still used to some extent, as, for instance, in Styria, where the hearths of furnaces smelting white forge-iron are built of blocks of serpentine, set with a mortar of fire-clay and magnesite. In Sweden the hearth and boshes are often lined with a mixture of crushed quartz and fire-clay, rammed round a core, giving the required profile. When fire-bricks are used, they are made as large as can be conveniently handled, the individual bricks being moulded in segments of the proper curvature of the ring. The thickness of the hearth wall is from 30 to 42 inches; that of the stack, from 30 inches at the bottom to 2 feet at the top.

Fire-bricks for blast furnaces require to be not only refractory, but to resist corrosion by slags, and must not wear too rapidly under the abrading action of the descending materials; these requirements are best met by a brick that is not too fine in texture nor too purely siliceous. The bricks in the highest repute at the present time are those made on the Tyne, and in the West of Scotland at Garnkirk and Glenboig, which are now largely used in foreign as well as in British iron-works.

For the hearth bottom blocks of strong sandstone, such as the conglomerates of the Millstone Grit or New Red Sandstone formations, are still used to some extent; but generally two or more layers of heavy fire-bricks, 2 to 3 feet thick, upon a bed of sand are preferred. The bricks are laid as inverted arches, or with wedge-shaped joints, with the smaller ends upwards, so that they may not be lifted in the event of molten iron working through the joints.

In an old furnace, such as that represented in figs. 46, 47, a heavy pyramid of masonry, formed by the union of the stack pillars, encloses the furnace proper, a hollow space being left between the stack and casing walls, which is filled with broken bricks or slags to allow for change of shape by expansion in the inner wall; but in all modern furnaces the circular form is adopted, and the amount of casing to the stack proper is very much reduced. The progress of construction in this direction will be seen by a comparison of figs. 50, 51, 52, and 54. The lighter furnaces are known as *cupola-furnaces*. They are either strengthened by numerous bands of wrought-iron, as in the woodcuts, held together by collars or screw-bolts, or are entirely cased with a jacket of wrought iron plates rivetted together, the latter construction being now very generally adopted. In all the examples given above, the casing wall is thickened at the top to give support to the charging platform or gallery at the throat of the furnace, but in the iron-cased forms the overhanging weight is carried on brackets rivetted to the shell-plates.

In Büttgenbach's furnace, which was originally constructed at Neuss in Prussia, and has since been adopted to some extent in other places, tubular pillars of cast- or wrought-iron are used to support the furnace top, so that the stack being entirely relieved of weight, no outer wall becomes necessary, and it may be constructed of a single thickness of bricks. Fig. 54 is a section of a furnace of this kind erected at Anzin. The tubes that support the tunnel-head platform serve also as gas-collectors or down-comers.

*Protection of the Hearth and Boshes.*—In modern blast-furnaces blown with highly heated air, the corrosive action of the fluxes and cinder upon the bricks is so considerable that special means are required to protect the lower part of the furnace. This is done by building into the brickwork of the hearth, at several levels, hollow cast- or wrought-iron boxes of the shape of the bricks, which are connected with each other, and with a water main, so that a current of cold water may be maintained through them. This chills a thin layer of slag upon the wall, and so protects it from being fluxed by the lime in descending. This class of construction is mainly used in furnaces smelting manganiferous ores. In America a complete water-casing or jacket is used in some cases to a height of 8 or 10 feet above the hearth bottom. The same effect is often obtained by keeping water flowing over the outside of the hearth wall, and for the same reason it is now customary to use much thinner walls than formerly, two feet being considered a sufficient thickness for the hearth brickwork.

The points of most active combustion, namely, those where the blast is introduced, are protected by the use of so-called water-tuyers, which are cooled by a continuous circulation of water. Figs. 48, 49, are an example of a water-tuyer used in Staffordshire; it may be made of wrought- or cast-iron, copper, or bronze. In Scotland the cooling-water

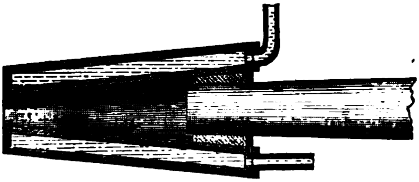

Fig. 48.—Water-Tuyer; longitudinal section.

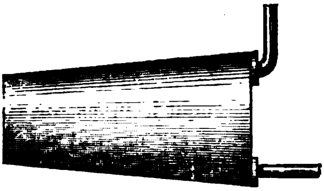

Fig. 49.—Water-Tuyer; side view.

is supplied by a wrought-iron pipe coiled round a conical mandrel, and enclosed in a mass of cast-iron. Lloyd's spray-tuyer differs from the older forms in being open at the back, and the cooling-water is driven in the form of spray from the end of a perforated pipe against the port of the tuyer. This has the advantage over the closed tuyer of not being liable to block up with deposit when using water containing lime or much suspended matter.

The tuyers, which are placed in the same horizontal plane, vary in number, according to the size of the furnace, from one or two in small charcoal stacks up to six or seven in the largest coke furnaces. When the number is large, they are usually laid with their axes pointing a little away from the centre line of the hearth. As a rule, they are horizontal, an inclined or plunging position being only adopted in the Styrian furnace, making white forge-pig with a minimum of carbon.

When working with very refractory slags, as in smelting ferro-manganese, the hearth bottom sometimes rises by the accretion of infusible masses below the tuyers, and the life of the furnace may be jeopardized if means are not provided for blowing at a higher level. In such cases, therefore, a row of blank tuyer-openings are built into the furnace above those ordinarily used, so that, when necessary, the wall may be broken through and blast-nozzles introduced. These are generally known in England by the name of *monkey-tuyers.*

The under side of the arch, covering the *fore-hearth*, is sometimes protected by a water-cooled casing which is known as the tymp.

This covering of the fore-hearth, which is bounded in front by the dam, is called the *tymp-arch*, and is, in large furnaces, protected by a casting, having within it a wrought-iron pipe through which a current of cold water is conducted, in order to protect the brickwork from intense heat and from the corrosive action of molten slag.

A semicircular depression on the top edge of the dam, known as the *cinder-notch*, forms a passage for the slags, which are often moulded into large blocks by being run into a shallow iron truck, provided with movable sides. When this *cinder-tub* has become full, it is removed and replaced by an empty one; as soon as it has sufficiently cooled to become solidified, the block of slag, which, in some cases, weighs several tons, is lifted from the waggon and thrown on the cinder-heap. In small furnaces, and particularly in those in which charcoal is the fuel employed, the front of the dam is formed into an inclined plane, or *cinder-fall*, on which the slag solidifies in thin layers, and may be readily removed by manual labour. In Staffordshire, slags are allowed to collect in a basin in the floor of the casting-house, called the *roughing-hole*, and when sufficiently consolidated are lifted, by means of a crane, upon a waggon and carried to the cinder-tip.

The *tap-hole* for withdrawing the molten iron from the furnace is in the form of a narrow vertical slot passing through the dam and dam-plate, and extending from the bottom of the hearth to a height of about eighteen inches above it. This is easily stopped by a packing of sand tightly rammed into it, and remains closed during the filling of the hearth, but is readily penetrated by a pointed iron bar at the time of casting.

The space between the top of the *dam* and the tymp-arch is also closed either by sand or by a temporary wall of fire-brick and clay, a small passage for the escape of slag being alone left open. Sometimes the level of the dam is raised above that of the bottom of the tymp, causing the metal in the hearth to be covered by a bath of molten slag, from which a stream flows continuously, as the fore-hearth is not stopped.

In many modern furnaces the open fore-hearth and dam are dispensed with, the hearth being closed all round, and a special tuyer provided for the removal of the slag. This, known as Lurman's slag-tuyer, is a water-tuyer of peculiar shape, with an aperture of about 2 inches diameter.

On either side of the tymp, cast-iron plates with vertical notches are frequently placed for the purpose of affording support to the heavy tools employed in clearing the hearth, and for other operations connected with the routine of working the furnace.

When the gases are allowed to burn freely at the throat of the furnace, it becomes necessary to provide a chimney to carry the flame clear of the charging place. This, called the *tunnel-head*, consists of a cylinder of brickwork from 8 to 12 feet in height, varying in diameter with the size of the furnace; it rests on the platform, and is strongly bound with wrought-iron. When the gases are collected for the purpose of being employed as fuel, the arrangements of the head of the furnace are of a more complicated nature. Some of these will be described when treating of the employment of waste gases.

Figs. 50–54 will serve to show the nature of the progressive changes which have, within the last thirty years, taken place in the construction of the blast-furnace in this country. Fig. 50 is a vertical section of a blast-furnace at Oldbury, built about 1853, blown with six tuyers. Fig. 51 is one of the older furnaces at the Stockton Iron-Works; and fig. 52, one of a pair erected at Ditton Brook, near Warrington, in 1872, which have closed tops, and from which the waste gases are drawn off to be used as fuel. Fig. 53 is a horizontal section of this furnace immediately below the level of the tuyers. It will be seen that the body of the masonry is supported by five brickwork pillars, *a*. Hot-blast is employed, heated by the waste gases, and introduced into the furnace through four tuyers.

When furnaces are slightly built and are closely hooped or entirely cased with iron, the superstructure is frequently supported on cast-iron standards, and they are entirely encased with boiler-plate to the throat.

In order that moisture may readily escape, and the brickwork be prevented from splitting, through the pressure of confined vapour, the masonry constituting the exterior casing of the older furnaces is often traversed by numerous small channels, by which the drying of the mass is facilitated. The work is strongly bound together, on the outside, by iron bands, which are made to bind tightly either by keys or by screws

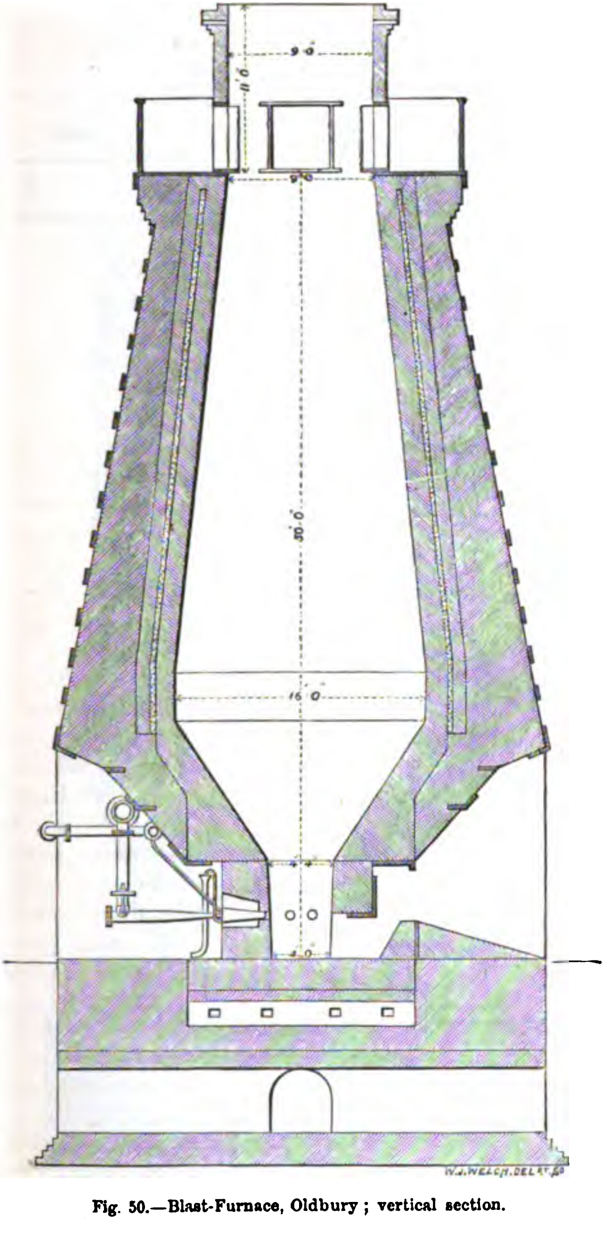

Fig. 50.—Blast-Furnace, Oldbury; vertical section.

and nuts. When the furnace is rectangular these bands are held together by long vertical bars, to which they are attached by loop-eyes or strong screw-bolts, and by this means great strength and solidity are communi-

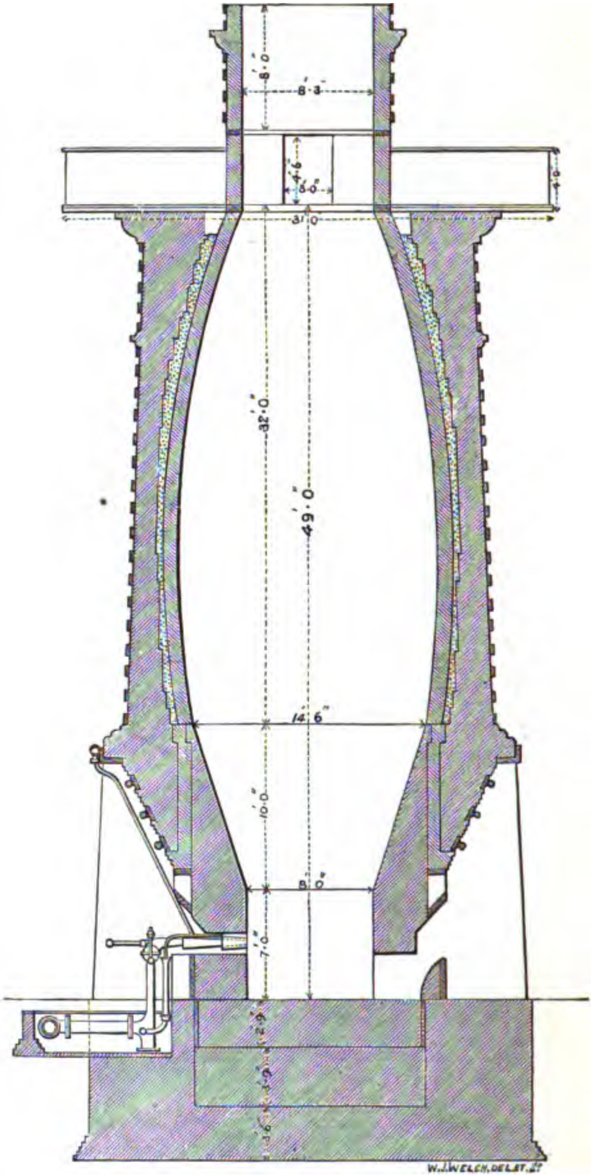

Fig. 51.—Blast-Furnace, Stockton ; vertical section.

cated to the structure. The dimensions of blast-furnaces differ very much, according to the period at which they were erected and the nature of the ores operated on. The height is extremely variable, some furnaces being

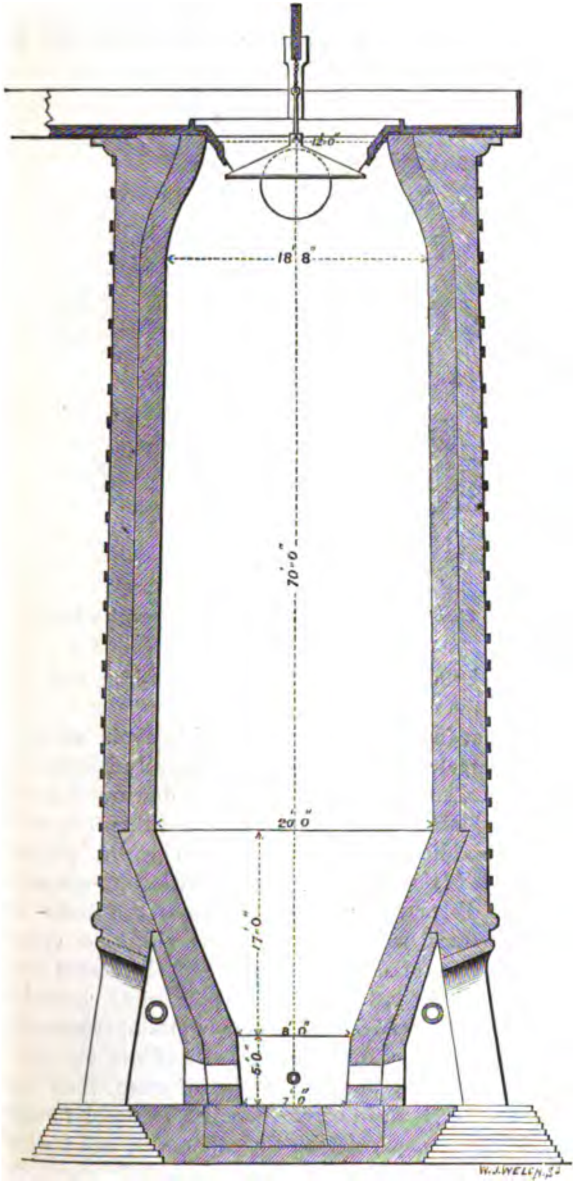

Fig. 52.—Blast-Furnace, Ditton Brook ; vertical section.

only about 30 feet high, including the chimney, whilst others reach an elevation of over 90 feet.

The most common height is, however, from 50 to 70 feet, exclusive of the chimney, which, when present, is from 8 to 12 feet in length, and is formed of radial bricks, bound by stout iron rings and girders; doorways are left in the sides for the introduction of ore and fuel. The throat is protected by a large annular plate of cast-iron, and on this rest the foundations of the chimney.

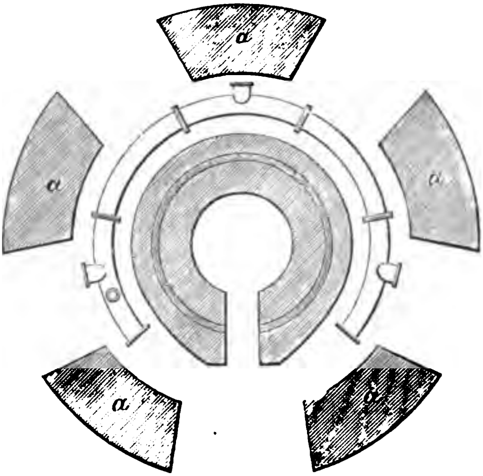

Fig. 53.—Blast-Furnace, Ditton Brook; section through hearth.

In building blast-furnaces it is now usual to avoid abruptly varying slopes, and the diameter is sometimes continuously increased from the throat to the boshes, and is thence contracted downward to the hearth bottom in a somewhat similar way. In Scotland this form of furnace is employed, with the addition of a wide cylindrical hearth. Slightly curved stacks, with conical boshes and cylindrical hearths, are almost universal in the Cleveland district, but in South Wales the boshes are often conical, while the stack, which is for a certain distance cylindrical, is terminated by a species of dome. The angle of the boshes is now made much steeper than formerly, following the practice of charcoal furnaces. In America, a bosh angle of 78° to 82° is now (1885) commonly adopted.

In French and German furnaces curved outlines are less common than in this country, and, in the majority of cases, their hearths are proportionately smaller than in ours. Swedish charcoal furnaces are of considerable height as compared with their diameter, and the hearth and boshes form part of the same cone, which is usually very acute. The stack is commonly cylindrical. Fig. 55, from Percy, is a vertical section of a furnace erected in 1857 at Sten, near Finspong. In all essential respects this furnace is similar to those constructed in England, and consists of an inner lining of fire-brick and an outer shell of less refractory material. The hot-blast stove is heated by the waste gases withdrawn

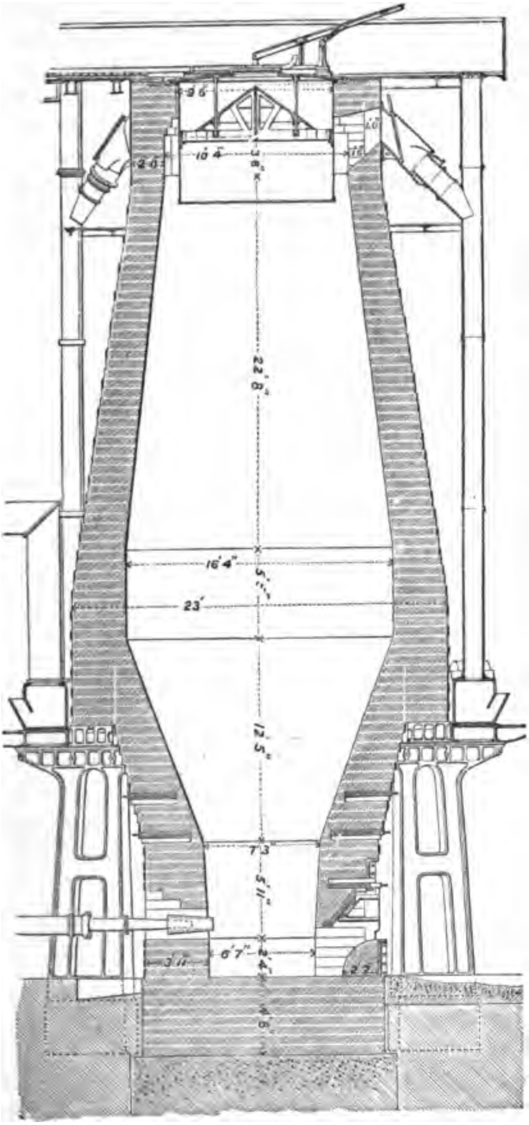

Fig. 54.—Büttgenbach's Blast-Furnace ; vertical section.

from the opening, *a*, in the upper part of the furnace, and the air-pipes are so arranged that, when necessary, the blast may enter the furnace without being heated by first passing through the stove. Nearly opposite the opening, *a*, is another, by which a portion of the gas is drawn off for the purpose of supplying fuel to the kiln in which the calcination of the ores is effected; the mouth of this furnace always remains open.

The height and other dimensions of blast-furnaces differ according to the nature of the ore treated and of the fuel employed, and no general rules can be laid down with regard to the form best suited for any particular class of ore. The most useful guide in the construction of a blast-furnace is afforded by the condition of other furnaces when they are blown out after working the same kind of ore under similar conditions.

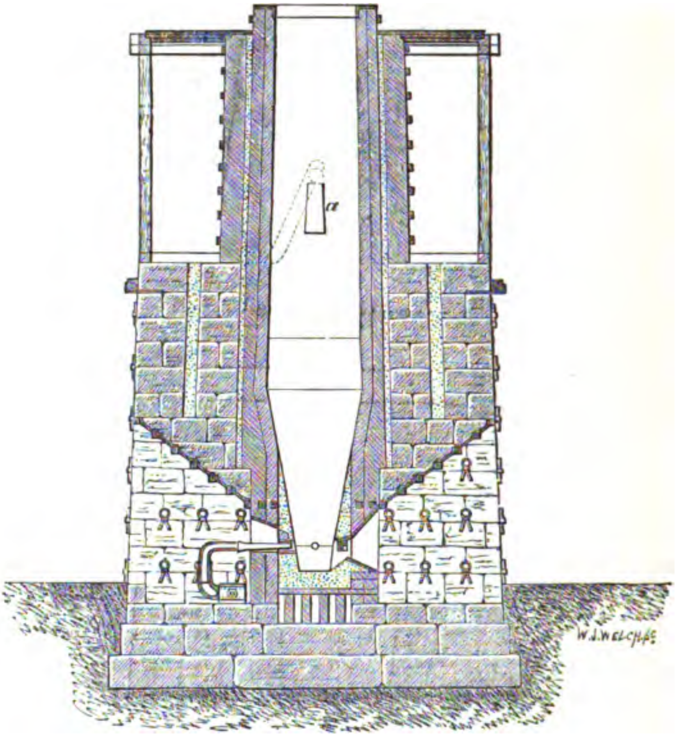

Fig. 55.—Swedish Charcoal Furnace; vertical section.

It is evident that, by constructing the various parts in accordance with the indications thus obtained, not only a certain amount of fuel may be economized, but the apparatus may be brought to its best working condition more quickly than in the case of the most suitable form having to be obtained by the erosive action of the slags.

An increased production of iron from a given burden can only be obtained by augmenting the smelting power of the furnace in which it is treated, since it is manifest that fresh charges can be introduced only in proportion to the rapidity with which those which have preceded them are removed. The power of fusion is mainly dependent on the rapidity with which fuel is consumed by the oxygen of the blast; and as the combustion of fuel is chiefly confined to the region of the tuyers, it follows that by augmenting the diameter of the hearth an enlarged area of active combustion is obtained.

The height of a blast-furnace should be mainly regulated by the character of the fuel employed, as regards its power of resisting the crushing action of a large number of charges forming a high column of materials; very high furnaces are consequently not used with tender fuel, and the favourable results obtained from the tall furnaces in the Cleveland district are, to a great extent, due to the exceedingly resistant nature of the coke employed as fuel. The height of furnaces in which anthracite is employed is not generally great, since the fuel is liable, by decrepitating, to cause obstructions, which are only to be avoided by the use of a more than usually powerful blast. Such furnaces are, therefore, generally low, wide, and blown by a considerable number of tuyers. It is of great importance that the dimensions of a furnace should be so regulated that the reduction of the ore may take place at a low temperature, as otherwise silicates of iron will result, with the formation of scouring slags and the production of white cast-iron. The harder a furnace is driven, all other conditions being the same, the greater will be the deterioration in the quality of the metal produced, caused by the more rapid descent of the charges; it consequently follows that to manufacture an increased quantity of iron, without injury to its quality, it is necessary to employ a larger furnace.

The following table, published by Mr. Gjers, in the 'Journal of the Iron and Steel Institute,' shows the progressive increase in size, during twenty years, of Cleveland blast-furnaces, giving their dimensions and capacity in the order of their respective dates.

DIMENSIONS OF BLAST-FURNACES.

| Date. | Name of Firm. | Furnaces. | Height. | Width of Boshes. | Capacity. |
|---|---|---|---|---|---|
| | | No. | Feet. | Feet. | Cubic feet |
| 1851 | Bolckow & Vaughan . . . | 3 | 42 | 15 | 4,566 |
| 1853 | Bell Brothers . . . . | 6 | 47½ | 16½ | 6,174 |
| „ | Gilkes, Wilson, Pease & Co. . | 2 | 45½ | 14½ | 5,100 |
| „ | Bolckow & Vaughan . . . | 6 | 54 | 15 | 7,166 |
| 1854 | Cochrane & Co. . . . . | 4 | 55 | 16 | 7,175 |
| „ | B. Samuelson & Co. . . . | 3 | 50 | 14 | 5,050 |
| „ | Bolckow & Vaughan . . . | 3 | 54 | 15 | 7,116 |
| „ | Gilkes, Wilson, Pease & Co. . | 2 | 55 | 14½ | 6,800 |
| 1856 | Stockton Furnace Company . . | 3 | 50 | 16 | 6,341 |
| „ | Norton Iron Company . . . | 3 | 50 | 15 | 6,000 |
| 1858 | Thomas Vaughan . . . . | 6 | 56½ | 16 | 7.000 |
| „ | Hopkins, Gilkes & Co. . . . | 2 | 56 | 16 | 7,200 |
| „ | Jones, Dunning & Co. . . . | 2 | 58 | 17 | 8,000 |
| „ | Bolckow & Vaughan . . . | 1 | 61 | 16½ | 7,960 |
| 1861 | Gilkes, Wilson, Pease & Co. . | 1 | 55 | 16 | 7,700 |
| „ | Whitwell & Co. . . . . | 3 | 60 | 20 | 12,778 |
| 1862 | Bolckow & Vaughan . . . | 2 | 75 | 16½ | 11.985 |
| 1864 | B. Samuelson & Co. . . . | 4 | 69 | 20 | 15,500 |
| „ | Thomas Vaughan . . . | 3 | 70 | 18 | 12,000 |
| „ | Lloyd & Co. . . . . . | 4 | 67 | 20 | 15,000 |
| „ | Thomas Vaughan. . . . | 6 | 81 | 19 | 16,000 |
| „ | Stevenson, Jacques & Co. . . | 3 | 70 | 22½ | 17,000 |
| 1865 | Gilkes, Wilson, Pease & Co. . . | 2 | 75 | 21 | 17,700 |
| „ | Bell Brothers . . . . . | 2 | 80 | 20½ | 15,500 |
| „ | Bolckow & Vaughan . . . | 2 | 95½ | 16 | 15,050 |
| 1866 | Bolckow & Vaughan . . . | 1 | 75 | 20 | 12,972 |
| „ | Hopkins, Gilkes & Co. . . . | 2 | 75 | 24 | 20,000 |
| „ | Swan, Coates & Co. . . . | 2 | 75 | 20 | 16,090 |
| „ | Bell Brothers . . . . | 2 | 80 | 17 | 11.500 |
| 1867 | Norton Iron Company . . . | 2 | 85 | 25 | 26,000 |
| „ | Cochrane & Co. . . . . | 2 | 76 | 23 | 20.624 |
| 1868 | Gilkes, Wilson, Pease & Co. . . | 1 | 75 | 24 | 22,500 |
| „ | Stevenson, Jacques & Co. . . | 1 | 70 | 23½ | 18,000 |
| „ | B. Samuelson & Co. . . . | 1 | 69 | 21½ | 16,000 |
| „ | Lloyd & Co. . . . . . | 2 | 80 | 21½ | 18,000 |
| „ | Jones, Dunning, & Co. . . . | 3 | 73 | 18 | 12.000 |
| „ | Bolckow, Vaughan & Co. . . | 2 | 95½ | 22 | 25,940 |
| „ | Bolckow, Vaughan & Co. . . | 1 | 95½ | 23 | 28.800 |
| 1869 | Thomas Vaughan. . . . | 3 | 85 | 25 | 26,000 |
| 1870 | Bell Brothers . . . . | 4 | 80 | 25 | 25,000 |
| „ | Stockton Furnace Company . . | 2 | 80 | 24 | 24,613 |
| „ | Swan, Coates & Co. . . . | 1 | 75 | 23⅔ | 22.229 |
| „ | Cochrane & Co. . . . . | 2 | 90 | 30 | 41,149 |
| „ | Gilkes, Wilson, Pease & Co. . | 2 | 85 | 27 | 32,000 |
| „ | B. Samuelson & Co. . . . | 2 | 85 | 28 | 30,000 |
| 1871 | Gjers, Mills & Co. . . . | 2 | 85 | 25 | 26,000 |
| „ | Lackenby Iron Company . . | 2 | 85½ | 25½ | 26,676 |
| „ | Bolckow, Vaughan & Co. . . | 2 | 95½ | 24 | 28,950 |

**BLOWING-MACHINERY.**—The blowing-machine ordinarily employed, fig. 56, consists of a large cast-iron cylinder, A, accurately turned on the inside, and provided with a piston, P, made air-tight by a packing often consisting of tressed hemp. The cylinder is closed at both extremities by iron ends, and on the cover is a stuffing-box, through which passes the rod, R, connected with the piston. The cover of the cylinder is provided with openings communicating with the outer air, and furnished with valves, $v$, opening towards the inside. Another valve, $v'$, on the contrary, opens outwards, and communicates with a lateral chamber, B, also of cast-iron. The lower end of the cylinder is provided with similar apertures and valves, those marked $v$, which establish a communication between the external air and the space beneath the piston, open inwards, whilst the opening communicating with the lateral chamber is closed by a valve, $v'$, shutting in an opposite direction.

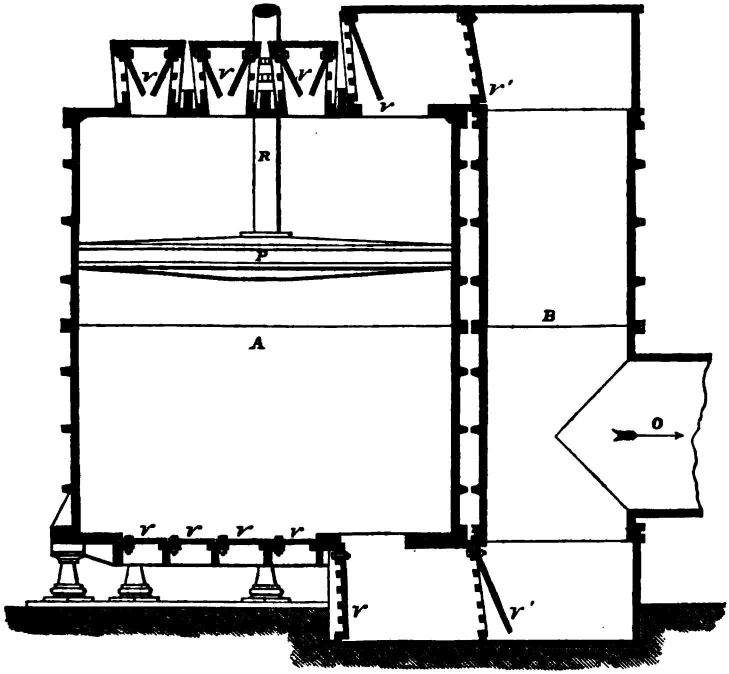

Fig. 56.—Blast-Cylinder, Dowlais; vertical section.

The better to understand the action of this apparatus, let us suppose that the piston has been raised to its full height in the cylinder, and has begun to be again forced down. If the valves $v$ are closed, the air contained in the upper part of the vessel gradually becomes more and more rarefied, and the difference of density between the air in this part of the cylinder, and that of the blast in the chamber, B, will cause the valve $v'$ to apply itself firmly against the metallic surface before which it is hung. The valves $v$, on the contrary, which open inwards, will be lifted as soon as the difference between the density of the enclosed air and that of the atmosphere is sufficiently great to overcome the resistance caused by their mechanical adjustments; and in proportion as the piston descends, the space behind it will be occupied by a supply of atmospheric air arriving from without.

The motion which causes the air above the piston to dilate, will evidently at the same time compress that which is beneath, in proportion as it approaches the bottom of the cylinder, and cause the lower valves *v*, opening inwards, to close firmly against the polished metal surfaces to which they are attached; whilst that marked *v'*, hung in a contrary direction, will open and allow the air to pass into the chamber, B, whence it escapes, through the aperture, O, to the pipes connected with the tuyers of the furnaces. In this way the upper portion of the cylinder draws the air from without during the descent of the piston, and forces that which is beneath it through the chamber into the pipes with which it is connected. When the piston is raised, the reverse of this takes place: the lower portion receives air from without, whilst the upper discharges that which it contains through the pipes leading to the tuyers. The machine is by this means made to throw into the furnace a nearly continuous flow of air; the only time at which the current is interrupted being that at which the piston has reached the full extent of its stroke, and before it has begun to move in a contrary direction.

As, however, it is of importance that the regularity of the blast should be maintained, the pipe, O, leading from the chamber, B, is made to communicate with a closed reservoir of wrought-iron, where the variations referred to are lost through the elasticity of the air itself. The piston of the blowing machine is now almost invariably worked by steam power, being often attached by a parallel adjustment to the oscillating beam of an engine. In some cases each machine is provided with two blowing cylinders acting alternately at each stroke made by the beam, by which the motion is communicated. The power required to work an apparatus of this kind necessarily depends on its size, and also on that of the furnace or series of furnaces which it supplies.

*Blowing-Engine at Dowlais Iron-Works.*—The large blowing-engine at the Dowlais Iron-Works, a section of the air cylinder of which is given fig. 56, was erected in 1851 by Mr. Truran, and has been described by Mr. Menelaus in the 'Transactions of the Institute of Mechanical Engineers.' Fig. 57 is a side elevation of this engine. The blowing-cylinder, A, is 144 inches in diameter, with a stroke of 12 feet, making 20 double-strokes per minute, the pressure of the blast being 3¼ lbs. per square inch.

The discharge-pipe, O, is 5 feet in diameter, and about 140 yards in length; thus answering the purpose of a regulator. The area of the entrance air-valves is 56 square feet. The amount of air discharged per minute, at the above pressure, is about 44,000 cubic feet.

The steam-cylinder, C, is 55 inches in diameter, has a stroke of 13 feet, with a steam pressure of 60 lbs. on the square inch, and works up to 650-horse power. Steam is cut off when the piston has made one-third of its course. There is also on one side of the steam-nozzle a small separate slide-valve for moving the engine by hand when starting. The cylinder-ports are 24 inches wide by 5 inches in depth, and the slide-valve has a stroke of 11 inches with a lap of half an inch. This engine is non-condensing, and the exhaust-steam is discharged into a cylindrical heating-tank 7 feet in diameter and 36 feet in length, containing the water employed for feeding the boilers; beneath the steam-cylinder there are about 75 tons of cast-iron framing, and 10,000 cubic feet of masonry.

The beam is cast in two parts, each weighing about 16½ tons, the total weight upon the gudgeons being 44 tons. It is 40 feet 1 inch from outside centre to outside centre, and is connected to the crank on the fly-wheel shaft by an oak sweep-rod, strengthened from end to end by wrought-iron straps. The fly-wheel, D, is 22 feet in diameter, and weighs 35 tons. Steam is supplied by eight Cornish boilers, each 42 feet long and 7 feet in diameter, with a single internal flue, 4 feet in diameter, in which there is a fire-place 9 feet long.

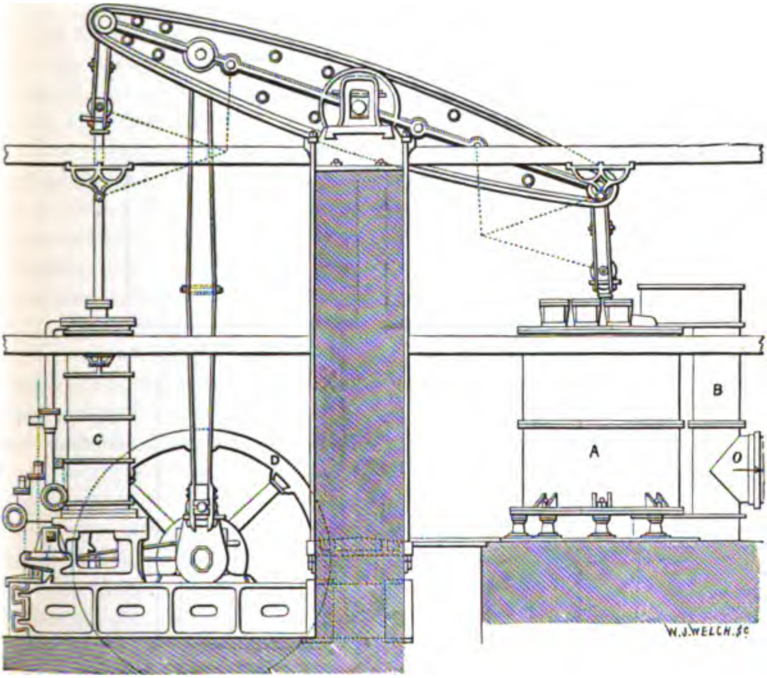

Fig. 57.—Blowing-Engine, Dowlais.

For some time this engine supplied blast to eight large furnaces, varying in diameter from 16 to 18 feet at the boshes; it is now, in conjunction with three other engines of smaller size, blowing twelve furnaces, some of which make upwards of 235 tons of good forge pig-iron per week; the weekly make of the twelve furnaces is about 2,000 tons of forge-pig.

With the exception of the cylinders, which were made and fitted at the Perran Foundry, Cornwall, the engine and boilers were made at the Dowlais Works, under the superintendence of the Company's engineer.

*Blowing-Engines in the North of England.*—The blowing-engines employed in the North of England are often of vertical construction, and are sometimes coupled in pairs, having a fly-wheel between them, with cranks at right angles, as in fig. 58, which represents the arrangement

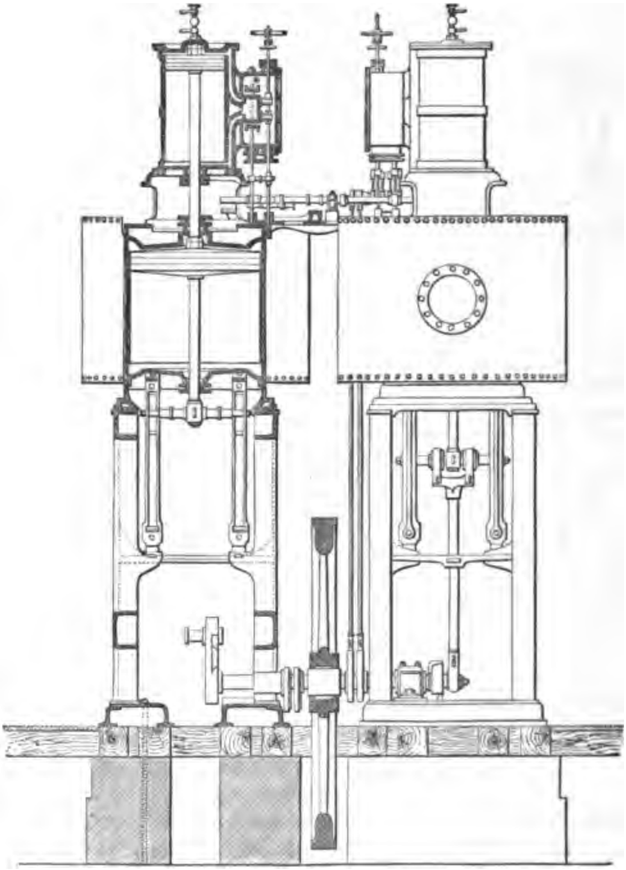

Fig. 58.—Blowing-Engine, Newport.

employed at Newport, near Middlesborough, described by Sir B. Samuelson (May 1871), in a paper read before the Institute of Civil Engineers.

The substitution of slide-valves driven by eccentrics for clack-valves, for the admission and discharge of the air from blast-cylinders, has been tried at different times; but such engines have not been found advantageous, owing to the large amount of friction on the valve-surfaces, and the great wear and tear to which, from the rapid motion, the machinery is subjected.

In Austria small direct-acting blast-engines, having the steam-cylinder uppermost, are much used for charcoal furnaces; they are generally of small dimensions, averaging from 25- to 30-horse power, and deliver from 2,300 to 2,500 cubic feet of air per minute.

Horizontal blast-engines are often preferred in Rhenish Prussia; the cylinders are placed on the same line, and the rod which carries the piston passes through both covers of the blast-cylinder, and runs in guides on either side. Two engines of this description are not unfrequently coupled, but they are so constructed that one of them may be readily thrown off in case it should not be required. An engine of from 30- to 40-horse power is sufficient for blowing an ordinary charcoal furnace, but a single furnace working with coke requires a blowing-engine of from 90- to 100-horse power.

The pressure of the blast varies with the nature of the fuel employed and the burden of the furnace. In some parts of Europe the pressure of air employed for charcoal furnaces does not exceed $\frac{3}{4}$ inch of mercury, while in American anthracite furnaces a pressure of 15 inches, corresponding to $7\frac{1}{2}$ lbs. per square inch, is often used. In this country, with tender fuel, a pressure of from $2\frac{1}{2}$ to 3 lbs. is employed, but with hard coke it ranges from $3\frac{1}{2}$ to 5 lbs. per square inch.

The practice of blowing several furnaces with one engine, although mechanically economical, is attended with considerable risk should any break-down of the machinery take place. It is, therefore, always desirable that there should be a reserve of blowing power, and that the work should be so distributed between two or more machines, that in case of an accident to one of them the blast may still be efficiently kept up. When there is but one furnace, a pair of coupled engines, capable of being worked independently of each other, may be employed with advantage.

The steam power required for furnishing the blast to a modern furnace of large size is estimated at $2\frac{1}{2}$-horse power per ton of coke burnt per twenty-four hours. Thus a furnace making 90 tons daily with a coke consumption of 21 cwts. requires $2{\cdot}5 \times 1{\cdot}05 \times 90 = 236\frac{1}{4}$-horse power. The large American furnaces, making 200 tons and upwards per day, take as much as 500-horse power to blow them, which is usually distributed over several small engines.

On account of the variations of pressure at the different parts of the stroke and the pulsations caused by the reciprocating action of the piston, the blast issues from the blowing-cylinder with a somewhat irregular flow. In order, therefore, to obtain a steady blast from the various nozzles, it becomes necessary to employ some means for rendering the pressure constant. This may be effected either by receiving the blast in a reservoir having a capacity several times greater than that of the blowing-cylinder, or by delivering it into a second cylinder provided with a loaded piston, which rises when the amount of blast increases, but falls and exercises a compressing action when the supply of air is temporarily diminished.

The same result may be obtained by the use of a loaded bell floating, like a gas-holder, in water; but fixed reservoirs of sufficient capacity are equally efficient, and are now generally preferred. These are usually made of sheet-iron, and had formerly a spherical or dome-like form, but they are now more frequently cylindrical, and should have a capacity from forty to fifty times greater than the volume of air delivered per second by the blowing-engine. When, however, the blast-main is long and of considerable diameter, with two or more engines blowing into it at the same time, it frequently happens that sufficient uniformity of pressure can be obtained without the use of a special regulator. In some of the old iron-works a brickwork chamber, lined with cement, was used as a regulator for the blast.

HOT-BLAST.—A patent was granted in 1828 to Mr. James Beaumont Neilson entitled 'Improved Application of Air to produce Heat in Fires, Forges, and Furnaces, where Bellows or other blowing Apparatus are required.' There is reason to believe that the patentee had originally no just conception of the great value of his invention, or of the important influence it was destined to exert on the manufacture of iron. The particular reference made by him in his specification to smiths' fires and iron-founders' cupolas, would seem to indicate that he regarded his invention as being more particularly applicable to such purposes than to the blast-furnace.

Mr. Neilson, and others with whom he had entered into partnership, granted a license in 1832 to Messrs. Baird, the proprietors of the Gartsherrie Iron-Works, Scotland, in consideration of receiving one shilling per ton on the iron produced at their establishment. This payment was subsequently disputed by the licensees, principally on the ground of insufficient description and want of novelty, but they further contended that cold-blast was practically more economical. The trial took place at Edinburgh in 1843, when the jury awarded the patentees damages to the amount of £11,867, 16s. The value of this invention, which, at a comparatively recent date, was thus disputed, is now universally admitted, and the employment of the hot-blast has been proved not only to be attended with a great economy of fuel, but at the same time has been found to increase the productive power of the furnace. Heated air is consequently at the present time employed to the almost total exclusion of cold-blast, in all the principal iron-producing districts of the world; cold-blast being retained only for the production of certain special brands of cast-iron which command a high price, and may therefore be manufactured at a correspondingly enhanced cost.

The temperature to which the blast may be advantageously raised appears to be limited only by the wear and tear of the apparatus and by the difficulty of keeping it tight when the air is very strongly heated. The blast is not generally heated beyond 350° or 400° C., but it is found that by using air heated to 650° instead of 400° C., a saving of 5 cwts. of coke per ton of iron made can be effected. In some cases the blast is now used at a visibly red heat, or about 600° C., but when such extreme temperatures are employed the rapid destruction of the metallic pipes of the stove renders a special construction of the heating apparatus necessary.

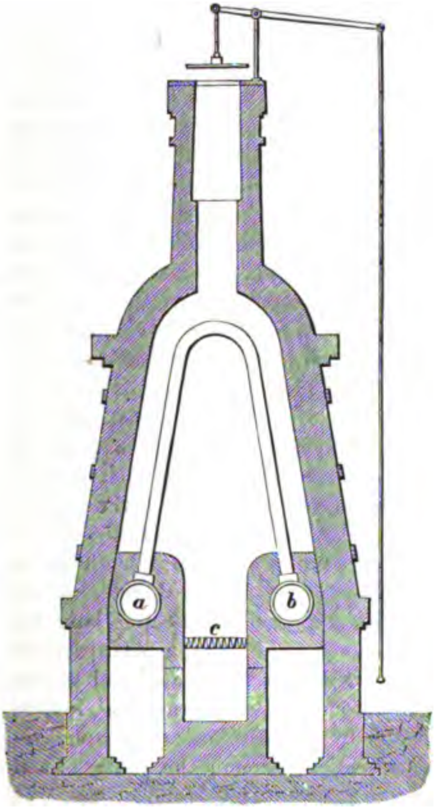

Fig. 59.—Common Stove; transverse section

*Common Stove.*—An apparatus commonly employed for producing hot-blast of comparatively low temperature consists of a series of parallel tubes, arranged in a fire-brick chamber, where they are heated externally, either by the combustion of solid fuel, or by that of the waste gases from the furnace. One end of these tubes is in communication with a main, which supplies cold air from the blowing-engine, while the other is connected with that which conveys heated air to the tuyers. In the older stoves the fire-place is rectangular, and two mains, which are parallel to the longer sides and circular in section, are provided with a number of sockets into which the ends of the heating-pipes are cemented.

These pipes, similar in section, and having the form of a syphon or arch, are placed in the position shown in fig. 59, which represents a transverse section of a hot-blast stove; their extremities are severally attached to the mains, *a* and *b*, by cement joints. The grate, *c*, extends along the whole length of the apparatus, and the flame and heated gases, after playing against the under sides of the tubes, pass around and between them, finally escaping to a chimney provided for that purpose. The cold air from the blast-engine, entering by the main, *a*, on one side, flows continuously through the arched pipes, where it becomes heated, and passes off to the tuyers by the opposite main, *b*. In order to obtain a larger amount of heating surface, the arched or horse-shoe pipes are usually made with a flattened elliptical section instead of a circular one, and inverted V-shaped pipes are frequently employed instead of those of the arched form shown in the engraving. The heating power of the apparatus has also been augmented by the introduction of stops in the mains, by which the air is compelled to pass alternately backwards and forwards through the vertical pipes before being conducted to the furnaces. In all cases the cold air is introduced at one end of the stove, and passes off to the tuyers from the other extremity. The arched pipes of hot-blast stoves are liable to become broken by the expansion and contraction caused by variations of temperature, unless they are allowed a certain freedom of motion; this is provided for by supporting one of the mains on rollers in such a way as to admit of its moving inwards or outwards, according as the pipes either contract or expand.

*Circular Stove.*—Round stoves are sometimes employed in place of the rectangular stove above described, and, in such cases, the air-mains are replaced by an annular cast-iron box, having a square or trapezoidal section; this is divided by a central partition into two hollow rings, one of which corresponds to the cold and the other to the hot-air main of the ordinary stove. The vertical pipes, instead of being arched, are connected at top by a short horizontal connecting-piece. Fig. 60 represents a vertical section through the centre of a stove of this kind. Fig. 61 is a horizontal section through the air-box.

The cold air first enters the outer ring, *a*, where it is interrupted by a stop, fig. 61, and then reaches the inner ring, *b*, by ascending a number of the outer vertical pipes and descending an equal number of those of the inside series. A stop in the inner ring causes the air to again pass through an equal number of the vertical tubes into the outer one, by which its temperature is still further augmented.

*Pistol-Pipe Stove.*—Another modification, known as the "pistol-pipe stove," is made use of in some districts in this country, as well as in France, Germany, and other parts of the Continent of Europe.

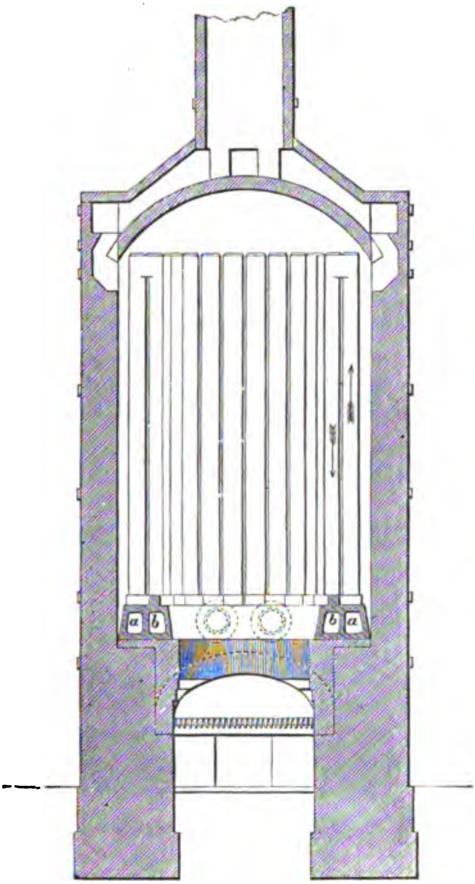

Fig. 60.—Circular Stove; vertical section

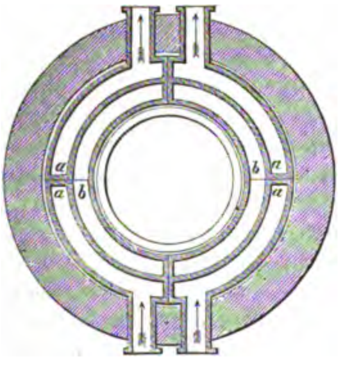

Fig. 61.—Circular Stove; section through air-box.

In this case the two vertical pipes or limbs are replaced by a single one divided longitudinally by a division reaching nearly to the top, which is closed, enlarged, and bent over in the form of a pistol-stock. These pipes are arranged on either side of a fire-place, as shown, fig. 62, and the cold air which enters one division descends through the other, and after becoming heated by the furnace, finally passes off to the tuyers.

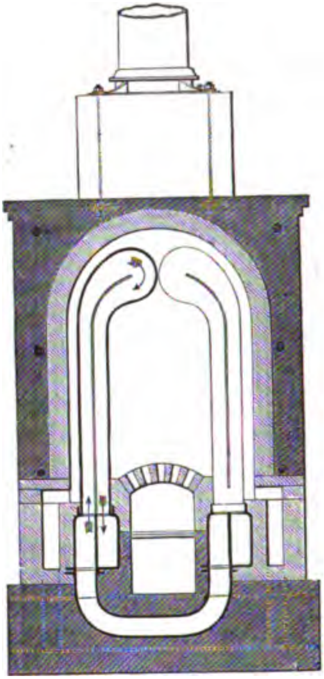

Fig. 62.—Pistol-Pipe Stove; transverse section.

*Stove used at Neustadt.*—At the Neustadt Iron-Works, Hanover, a form of stove is employed of which fig. 63 represents a vertical section. The heating coil consists of four cast-iron pipes united by semicircular bends, and three such series are so connected by branch pipes that the whole apparatus consists of twelve tubes. The cold air enters at *a*, and passing downwards, finally passes off in a heated state at *b*; the fuel employed is waste gas from the blast-furnace, which arrives through the large wrought-iron main, *c*, and is supplied to the furnace through jet-pipes, *d*. The jets, *d*, are provided with a central tube, through which air is admitted at *e*, in order to effect the combustion of the gas.

The pipe, *f*, is one of a series employed for superheating steam. This type of stove has been much used in North Germany, where it is known as the Westphalian pattern. The pipes generally used are of a flattened elliptical section, placed with the longer axes upright; the coil is sometimes 10 or 12 pipes high, and the bends, though not heated, are enclosed between walls. Another form, known as the Lorraine stove, has the bends on one side only, the pipes being closed at the back and divided by a mid-feather, as in the pistol-pipe stove.

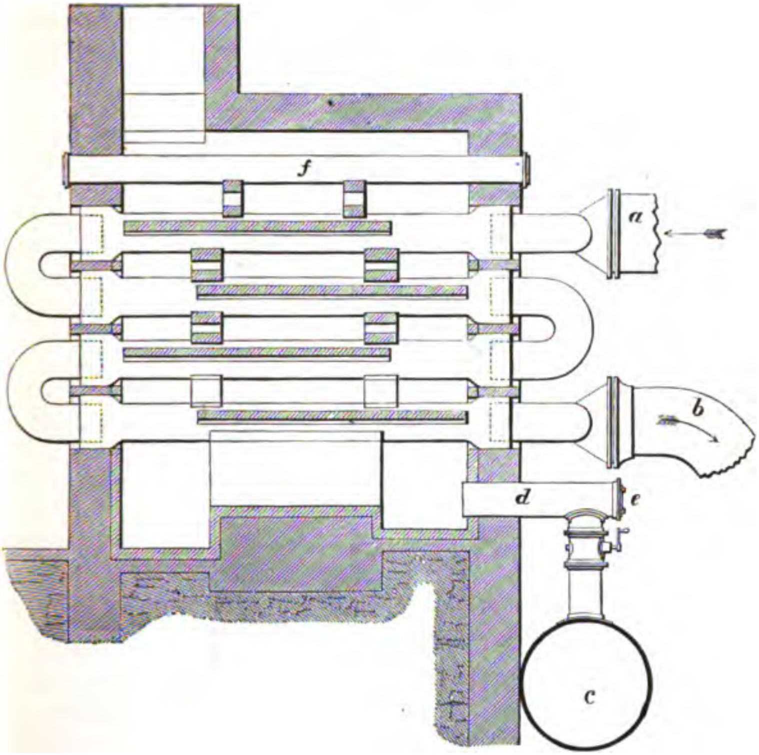

Fig. 63.—Hot-Blast Stove, Neustadt; vertical section.

*Cowper's Stove.*—When it is desired to heat the blast above a visibly red heat (about 1,100° F.), stoves containing large masses of fire-brick, which are worked intermittently on Siemens' regenerative principle, are substituted for those with cast-iron pipes. This important improvement in blast-furnace practice was invented by Mr. E. A. Cowper, whose stove is represented with its newest modifications in figs. 64, 65, 66. It consists of a cylindrical tower with a domed top, B, 40 to 65 feet high, and 26 to 28 feet in diameter, built of fire-brick, with an iron casing, I. A tube or flame-flue, F, placed eccentrically within the stove, extends from the bottom to the spring of the dome, while the remaining space being filled with fire-bricks forms the regenerator, R. The bricks are carried upon grids, supported by girders, P, so as to leave a clear space at the bottom, which is accessible by the manholes, M. The gas from the furnace, introduced by the gas-valve, G, to the burners, N, is burnt by air supplied through the air-valve, A, forming a body of flame in F, which descends through the passages of the regenerator and passes out by the chimney valve, V. The brickwork of the regenerator absorbs heat continuously from the flame until it reaches a maximum temperature at the top, when the pile is visibly red hot to within a few feet of the bottom. The gas is then shut off, and cold-blast, admitted by the valve, C, is passed through in the reverse direc-

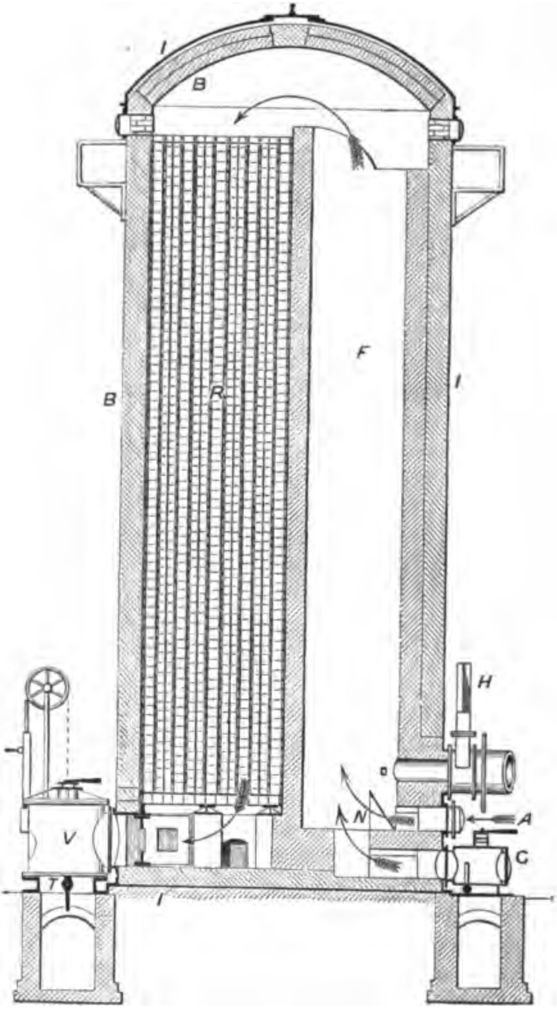

Fig. 64.—Cowper's Stove; vertical section.

tion, which becoming heated in its passage, cools down the brickwork in a corresponding degree, and is delivered as hot-blast by the valve, H. When the brickwork is cooled below a red heat for about three-quarters of its height, the blast is stopped and gas is turned on to re-heat it, and so on. Two stoves at least are provided for each furnace, one of which is on gas while the other is on blast. The regenerator is filled with Cowper's honeycomb bricks, fig. 66. These are hexagonal tubular bricks two inches thick with radial spurs at the angles, which when piled in the stove, as in fig. 64, give a series of six-sided flues with walls uniformly two inches thick. The top course is made with a sharp edge to check the lodging of any dust. The ordinary way of cleaning the stoves from dust that may have hung against the sides of the passages is to fill it with blast at the maximum pressure, and suddenly to blow it out by opening the throttle-valve, T, or by firing a gun in the space below the girders. When cold, the flues may be brushed down from above from the space under the dome, which is accessible from the gallery and cleaning-doors at the top. The internal angles of the honeycomb are rounded, affording a passage which is found to be less liable to choke from deposit than that formed of the thin bricks placed on edge, formerly used. At the present time about 370 of these stoves are used in the different iron-making countries of the world; the maximum temperature of blast obtained being from 1,500° to 1,600° F.[1] The average saving of fuel over pipe-stoves is about 5 cwts. per ton of iron made, while the make is increased about 20 per cent. The consumption of coke has been brought down to

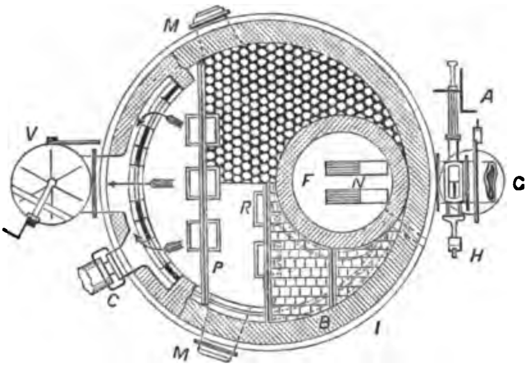

Fig. 65.—Cowper's Stove; horizontal section.

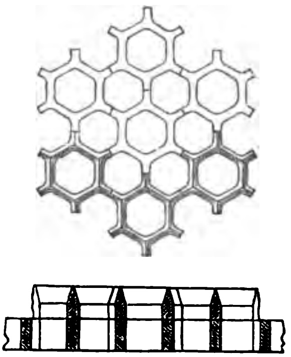

Fig. 66.—Cowper's Stove; details of brickwork.

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[1] The Cowper stoves at Newport Iron-Works, Middlesborough, each of 18,610 cubic feet capacity, of which 8,406 cubic feet are occupied by brickwork, leaving 10,204 cubic feet for gas- and air-passages, giving a heating surface of 42,860 square feet, furnish blast at 1,414° F. for a furnace 85 by 27 feet, making 461 tons of pig-iron per week from 38 per cent. ore, with a consumption of 21·8 cwts. of coke per ton of iron; or 2½ cwts. less than when iron-pipe stoves, giving a heat of 990° F., were used.

17 cwts. per ton of hæmatite-pig, 19 cwts. per ton of Cleveland pig, and with raw coal to 29 cwts. per ton of forge-pig, and 31 cwts. per ton of No. 1 pig.

*Whitwell's Stove.*—A modification of the regenerative stove introduced by the late Mr. Thomas Whitwell in 1865 is shown in its original form in figs. 67, 68, from which it will be seen that it is essentially a reproduction in fire-brick of the serpentine pipe-stove, with the coil placed vertically instead of horizontally. The gas admitted at A is burnt by air admitted through the feed passages, *a ;* the flame passing up and down by the serpentine passages between the numerous dividing walls, gives up its heat to the latter, which consist of 7-inch brickwork, and escapes

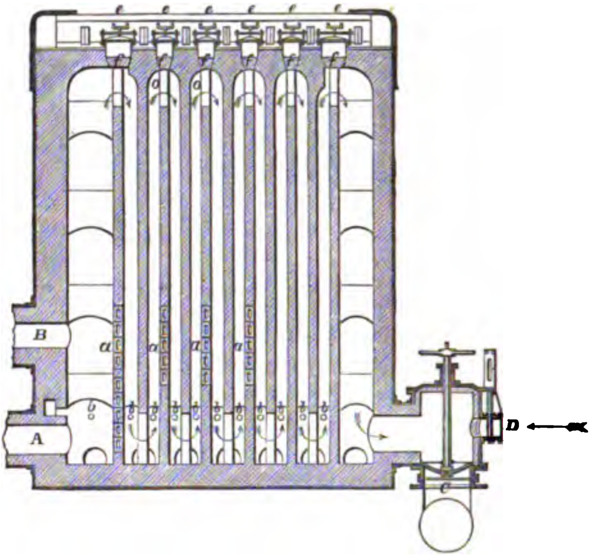

Fig. 67 —Whitwell's Stove ; vertical section.

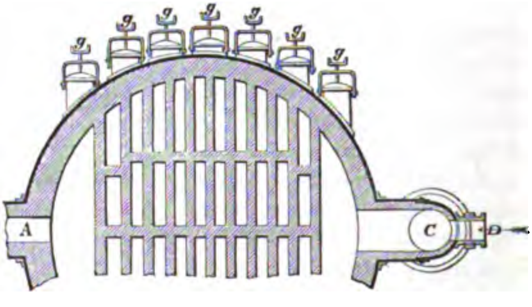

Fig. 68.—Whitwell's Stove ; horizontal section.

to the chimney by the passage C. When the stove is heated, cold-blast is admitted at D and passes out heated at B. The progress of the cooling is watched by sight-holes, *b*, the walls of the first two passages not being allowed to go below a red heat. The crowns of the arches at the ends *f* are removable, so that by opening the covers, *e*, the passages are made accessible for sweeping down or scraping the sides. The dust removed falls to the bottom of the stove, and is withdrawn through the side doors, *g*. The dimensions of the stove shown in figs. 67, 68, are 22 feet in diameter and 25 feet high, which size was used at the Consett Iron-Works in 1869. The flame passes twelve times through the entire height of the stove, the passages having a total length of 240 feet and a heating surface of 9,000 square feet. Subsequently the height has been greatly increased and the number of reversals of the flame correspondingly diminished, so that at present it is customary to build them of a height equal to or greater than that of the furnaces to which they are attached. The largest size adopted is 70 feet high and 25 feet in diameter, with 35,000 square feet of heating surface, the gas passing only four times through the stove. The thickness of the walls is also reduced to 5 inches, and a large number of thin walls are adopted, dividing the interior into numerous rectangular passages from 5 to 6 inches square. A domed top is also now used instead of the flat one shown in fig. 67.

A modification of the Whitwell stove known as the Gordon-Whitwell-Cowper stove has lately been introduced in America, in which the number of passages for the flames is reduced to three, and the exit of the spent gases is at the top under the dome, so that each stove can have its own chimney, and dispense with the long underground flues and tall stacks required when the gases pass out at the bottom of the stove.

The Massick and Crooke fire-brick stoves, which have been introduced at some Cumberland and Scotch furnaces, have a central combustion chamber with a number of narrow rectangular passages forming the heat-absorbing surface arranged concentrically around it. A central heating-tube or gas-burner was adopted in the early form of Cowper's stoves, as figured in the last edition of this work, but it was subsequently abandoned for an eccentric position, on account of the difficulty of equally distributing the gas through the whole of the regenerative passages.

*Blast-Pipes and Nozzles.*—The blast issuing from the stoves is carried round the furnace in a circular main, which, in the older ones, enclosed in a square casing of masonry, passes through the arched openings traversing the pillars supporting the stack, but in the newer form of furnaces is, at a certain height above the ground, secured to the columns enclosing the hearth. When the blast is at or above a red heat, the main is internally lined with firebrick. Opposite each tuyer a branch-pipe is brought down to the proper level; these are turned at right angles, and connected with the blast-nozzles. A throttle- or slide-valve is attached to each branch for the purpose of regulating or cutting off the blast, while a similar valve of larger dimensions is fitted to the main between the stove and furnace. A self-acting stop-valve is often used in the tuyer-pipe. This consists of a heavy flap-valve which opens outwards by the pressure of the blast, but falls back against its seat when the blast is stopped, and so prevents the penetration of furnace gas into the blast-main.

When cold-blast is employed, a conical nozzle is attached to the blast-pipe by a short leather tube, but when hot-blast is made use of, it is necessary that all the fittings should be of metal, and means are consequently provided for adjusting the nozzles by the aid of ball-and-socket joints and telescope tubes. Water-tuyers are made either of wrought or cast-iron, of a combination of both, or of copper or bronze; the latter are said to possess the advantage of not being readily destroyed by "ironing;" that is, of being attacked by the imperfectly fused masses of iron which sometimes adhere to them when the furnace is not in good working order.

The number of tuyers and the method of their arrangement vary in accordance with the size of the furnace and the nature of the fuel employed. Small charcoal furnaces have frequently only two tuyers placed on opposite sides of the hearth; three is, however, a more usual number, one being placed opposite the tymp, and the two others on opposite sides of the hearth. In the case of very large furnaces, the tuyers are sometimes arranged in series, two being placed on either side of the hearth, and the same number at the back, or three at the sides and either one or two at the back. Sometimes a special tuyer is added on the tymp side for the purpose of removing any obstruction caused by local cooling, and is only used in case of the hearth becoming obstructed by accumulations of imperfectly fused matter.

Utilization of Waste Gases.—Shortly after the application of hot-blast to iron-making, various attempts were made to employ the waste heat escaping from the throat of the furnace for the purpose of heating the air with which it is supplied.

One of the methods formerly employed for attaining this object consisted in ranging a series of iron pipes round the tunnel-head, in which the blast was heated by the flame passing out of the mouth of the furnace. In other instances the pipes were either coiled around the interior of the upper part of the stack, so as to be heated by direct contact with the ignited material which it contained, or were so enclosed in brickwork as to become heated by transmission. All these contrivances have, however, been successively abandoned, since from their inefficiency and the difficulty attending their repair when they got out of order, their use was not found advantageous.

An improvement on this plan was invented by Mr. James Palmer Budd, of the Ystalyfera Iron-Works in South Wales. Instead of making the heating apparatus an integral part of the furnace, the stoves were in this case so arranged as to allow of their being repaired without interfering with the action of the furnaces with which they were connected.

The stoves were built a little below the level of the throat of the furnace which they supplied with hot air, and a chimney, 25 feet higher than the top of the platform, afforded the means of drawing into them as much of the heated gases and flame as might be required. These were carried from the furnace by a series of flues about three feet below the top, communicating with the hot-air chamber in which were placed arched pipes heated by the gases issuing from the furnace. The chimney and its damper regulated the heat of the stove; cross-pipes connected the upright-pipes, and side-pipes conveyed the air arriving by mains to various cross-pipes. The heated air was afterwards conveyed to the tuyers by downcast-pipes. A door was placed in the brickwork of the building for the purpose of cooling the apparatus before entering to make repairs.

All these contrivances have, however, given place to various systems for conveying the combustible waste gases in pipes or culverts to the points where they are required to be burnt as fuel. In addition to the sensible heat which the gases are capable of directly communicating to any body with which they may be brought in contact, the whole of the gaseous column issuing from the throat of a blast-furnace is inflammable, even after its temperature has been reduced to that of the surrounding atmosphere. The combustion of these gases, therefore, affords a new and entirely distinct source of heat. Various patents have at different times been taken out for methods by which the heat thus lost has been sought to be usefully applied, but the difficulties attending the combustion of waste gases, added to the comparative cheapness of fuel, for a long time prevented their being extensively used in this country. On the Continent of Europe, where fuel is more expensive, the utilization of waste gases was much earlier introduced, but at the present time their employment has become almost universal in all iron-producing districts.

In many small charcoal furnaces, in which the throat remains open, the gases are taken off by iron pipes which perforate the brickwork from 10 to 12 feet below the top. In Sweden this plan is generally adopted, but it can be applied on only a limited scale, and the supply is liable to be somewhat irregular from the occasional partial stoppage of the openings by the descending charges.

Another method for collecting the gases is by partially closing the mouth of the furnace, so as to cause a slight impediment to the escape of its gaseous products, and then drawing them off by proper flues and tubes to where it is intended they shall be consumed.

In order to do this, a cylinder of cast-iron, of a smaller diameter than the throat of the furnace, and having a depth equal to its width, is sometimes used. This is suspended by a strong flange within the tunnel-head, and as the mouth of the furnace is constantly kept charged with mineral and fuel, while a clear annular space remains between the iron collar and the lining, it is evident that this must be filled with gases issuing from the apparatus, which may be readily conducted by flues or pipes to any situation where they may be required for combustion. In furnaces built especially with a view to economizing the heat to be obtained by burning the unconsumed gases, the internal iron lining is sometimes replaced by an annular flue made in the brickwork a few feet below the throat. This is connected by several openings with the interior of the stack, and as the charges thrown into the furnace above this point naturally offer a certain resistance to the exit of the escaping gases, they find their way into the annular flue before described, whence they are readily drawn off in any direction in which they are required, and may be conducted to a distance of several hundred feet.

*Method of Collecting Gases at Darlaston.*—Fig. 69 is a vertical section of the top of a furnace at Darlaston, where this system of collecting waste gases was introduced by Mr. G. Addenbroke. There are fifteen gas-openings, *a*, around the neck of the furnace, each 23½ inches wide and 11½ inches high, and consequently presenting an aperture of 270 square inches, making a total area of 4,050 square inches for drawing off the gases. The large gas-flue, *b*, surrounding the neck of the furnace is lined with fire-brick, and is 4 feet 3 inches high to the crown of the arch, having a mean width of 3 feet. The outside of the furnace from a little below the bottom of the flue upwards is cased with wrought-iron plates, to which is fastened a light iron gallery, *c*, for the convenience of cleaning the flue, *b*. A series of openings, *e*, is made all round the outer side of the flue, and these are closed by pieces of boiler-plate luted with fireclay, and held in their places by cross-bars and wedges; by means of these the whole of the flue may be cleaned out in the course of a few minutes, at any time when the blast is off the furnace. The bottom of the flue is at a lower level than the edge of the openings, *a*, in order that the dust carried over may accumulate for some time before in-

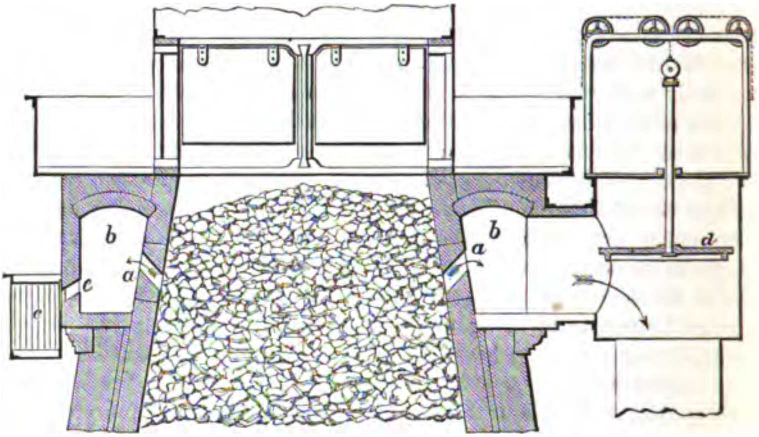

Fig. 69.—Top of Furnace, Darlaston; vertical section.

terfering with the exit of the gases. The gas-mains are 5 feet in diameter, and, in case of the top of the materials sinking below the gas-openings, any damage is prevented by shutting the valve, *d*, when the whole of the gas will burn at the mouth of the furnace, without injury resulting to the apparatus.

*Langen's Apparatus.*—When it is desired to utilize all the gases issuing from a furnace it becomes necessary to close the throat. At Siegburg, on the Rhine, Langen's apparatus for the collection of waste gases is employed; the furnace mouth being closed by a lid, in the form of a bell-shaped tube, resting in an inverted conical ring. This tube may be raised and lowered by means of a lever, for the purpose of charging, and is at its extremity provided with a lip, which dips into a water-trough in the gas-main, forming a perfectly air-tight joint. At the time of charging, the bell is lifted, and, sliding in the water-joint on the gas-tube, allows the charge in the cup-shaped ring to fall into the furnace. To prevent any accident from explosion, a safety-valve is placed on the top of the conical tube, and another on the gas-tube.

At Horde, Langen's apparatus has been modified as follows:—The mouth of the furnace, 9 feet in diameter, is closed by a flat lid of cast-iron, which, although it cannot be raised, may be readily turned on rollers, and is kept air-tight by a water-joint; a gas-pipe, 3 feet in diameter, is placed on this lid and is kept tight in the same way. This cover is provided, on its circumference, with four apertures closed by valves kept tight by water, through which, in quick succession, the charging of the furnace is effected. Before re-charging, the movable lid is made to traverse one-eighth of a revolution, thus uniformly distributing the materials round the circumference of the furnace.

*Cup and Cone.*—The simplest method of closing the throat of the furnace, and that which is most generally used, is the "cup and cone" or "bell and hopper," fig. 70, first applied by Mr. Parry at the Ebbw Vale Iron-Works. It consists of an inverted cast-iron cone, *a*, fixed to the top of the furnace, of which the lower aperture is about one-half the diameter of the throat. A cast-iron cone, *b*, is placed in the furnace below this cup or funnel, and suspended by a chain, *c*, to an arch-headed lever, *d*, carrying a counter-balance at the opposite end. The raising or lowering of this cone is often effected by a pinion on the shaft of a hand-wheel, *e*, gearing into a segmental rack attached to the lever. When the cone is raised, it bears against the bottom of the cup, and forms a stopper, which prevents the escape of gas from the top of the furnace. Thus prevented from escaping by the throat, the gaseous fuel is conducted through an orifice made in the wall of the furnace, above the level of the charges, and is conveyed, by iron pipes, to any part of the works where its combustion is to be effected.

In v. Hoff's modification of the cup and cone, which is largely used in Germany, the cone is surmounted by a cylinder which slides telescopically upon a fixed central tube like that of fig. 71, the union being made gas-tight by a water-joint. This combines the advantages of the central gas discharge and of the distributing cone.

*Method Employed at Grosmont.*—At Grosmont, Yorkshire, and Barrow-in-Furness, Lancashire, the waste gas is taken off in a wrought-iron tube, *a*, fig. 71, about 5 feet in diameter, which extends 5 feet down the throat of the furnace, and is lined on the inside and cased outside with refractory brick. This tube is supported by a brickwork dome, *b*, built in the throat of the furnace, supported by buttresses of the same material. This dome has six openings, *c*, at the sides, for charging purposes, and another in the centre, corresponding with the tube, *a*.

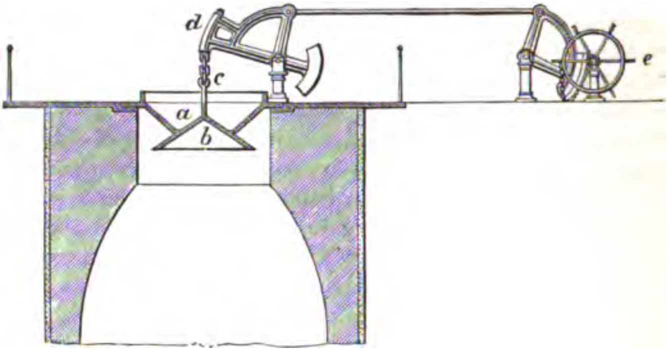

Fig. 70.—Cup and Cone; vertical section.

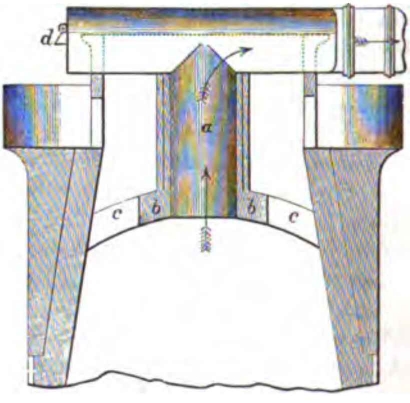

Fig. 71.—Top of Furnace, Grosmont; vertical section.

The furnace is provided with the usual brick chimney at top, which has wrought-iron swing doors corresponding with the openings in the crown. Expansion-boxes are fixed at intervals along the tube by which the gas is conducted to the boilers and hot-blast stoves, and a flap-valve, *d*, opening outwards, is placed at the end of the tube for the purpose of clearing it, and, if necessary, to act as a safety-valve. This form of gas-collecting by a central syphon tube was introduced by Mr. Darby, and is still used to some extent; but the supporting arches in the furnace are replaced by a girder-framing above the furnace top, from which the tube is suspended.

In some of the charcoal furnaces of America the charging is effected by barrows, which are constructed exactly on the principle of the cup and cone; and at Rhonitz, in Hungary, cylindrical charging barrows are employed, by which a portion of the material is dropped in the centre of the furnace whilst the remainder is distributed in a circle next the brickwork.

COMPOSITION OF WASTE GASES.—The composition of the gases of the blast-furnace at various heights has at different times been investigated by Bunsen, Playfair, Ebelmen, Scheerer, Tunner, and others. The results arrived at by these chemists have, after making due allowance for the different characteristics of the fuels employed, generally agreed very closely, and have afforded much valuable information relative to the chemical reactions which successively take place. The gases issuing from the throat of a furnace practically contain the whole of the carbon of the fuel consumed, with the exception of the comparatively small amount which has become fixed by the carburization of the metal.[1] This escaping carbon is chiefly in the form of carbonic anhydride and carbonic oxide gases, the oxygen of which has been principally derived from the air of the blast, but is to a less extent due to the reduction of oxide of iron. The whole of the nitrogen of the air blown in will also be present, together with small quantities of hydrogen and of hydrocarbons, which are to a great extent produced by the decomposition of watery vapour.

The following analyses give the composition of the gases issuing from various blast-furnaces:—

| Percentage by Volume. | 1. | 2. | 3. | 4. | 5. | 6. | 7. |
|---|---|---|---|---|---|---|---|
| $N$ . . . . | 62·34 | 57·22 | 55·62 | 56·64 | 55·35 | 54·91 | 52–60 |
| $CO_2$ . . . . | 8·77 | 12·01 | 12·59 | 11·39 | 7·77 | 18·36 | 3–8 |
| $CO$ . . . . | 24·20 | 24·65 | 25·24 | 28·93 | 25·97 | 26·66 | 25–30 |
| $CH_4$ . . . . | 3·36 | 0·93 | ... | ... | 3·75 | ... | ... |
| $C_2H_4$ . . . . | ... | ... | ... | ... | 0·43 | ... | 2–4 |
| $H$ . . . . | 1·33 | 5·19 | 6·55 | 3·04 | 6·73 | 0·07 | 5–7 |

1. Veckerhagen, Hesse Cassel; Bunsen; fuel, charcoal.
2. Clerval, France; Ebelmen; charge of brown hæmatite, limestone, and charcoal.
3. Audincourt, France; Ebelmen; charged with brown hæmatite, forge-cinder, limestone, wood, and charcoal.
4. Seraing, Belgium; Ebelmen; charge, brown hæmatite, mill-cinder, limestone, and coke.
5. Alfreton, Derbyshire; Bunsen and Playfair; charge composed of calcined argillaceous ores, limestone, and raw coal.

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[1] A certain amount of potassium cyanide is also produced in blast-furnaces.

6. Ormesby; smelting Cleveland ore with coke; temperature of blast 1,507 °F.; furnace 35,013 cubic feet capacity.

7. Average range of gases from Scotch blast-furnaces fed with splint-coal.

It will be observed that the nitrogen of the blast, which has passed through the furnace without taking any important part in the reactions which are continually going on, in each case constitutes more than one-half the entire volume of the gases evolved. The proportion of the nitrogen as compared with that of oxygen is, however, less than in atmospheric air, and as no appreciable absorption of this gas takes place in the furnace, it follows that the increase of from 12 to 18 per cent. in the amount of oxygen must be derived from the solid materials of the charge. This increase in the amount of oxygen is chiefly the result of the reduction of oxide of iron, and of the elimination of $CO_2$ from the limestone employed as flux. In certain cases, and particularly in that of hot-blast furnaces working on siliceous ores, a further but very small addition to the quantity of oxygen may be derived from the reduction of silica.

When furnaces are worked with raw coal, the gases evolved from them contain, in addition to the products of combustion, small quantities of various condensable vapours, especially tarry matter and ammonia. The recovery of these substances has latterly been commenced upon the large scale in the West of Scotland. At Gartsherrie the gases of eight furnaces, 50 × 15 feet, each consuming from 60 to 65 tons of coal in twenty-four hours, are led from the furnace top through a dust-catcher into an atmospheric condenser, like that of a gas-works, having 200 tubes 2½ feet in diameter and 40 feet high; this is followed by a water-condenser with 2,700 3-inch tubes 45 feet high, a first scrubber 80 × 25 feet, and a second scrubber 60 × 20 feet. The current through the different condensers is maintained by three sets of Root's blowers 6 feet in diameter, making 100 revolutions per minute. The products condensed are about 40 gallons of crude or 16 of boiled tar, and 22 to 23 lbs. of sulphate of ammonium per ton of coal consumed. The actual amount of ammonia contained in the gases is from 5 to 7½ lbs. per ton of coal, and is diffused through a volume of 125,000 cubic feet of gas. The amount of ammonia escaping the final condenser is estimated not to exceed 2 to 3 oz. of ammonium sulphate per ton of coal. The gases, when deprived of their condensable products, are distributed to the heating-stoves and boilers, and burnt in the ordinary way. The above is known as the Gartsherrie or Alexander & M'Cosh's process.

Dempster's process, another variety of the atmospheric method of condensing, has been established at Stoke-upon-Trent.

In Neilson's process the ammonia of the gases, after a preliminary cooling, is absorbed by subjecting it to a shower of dilute sulphuric acid in a lead-lined scrubber.

In Addie's process, used at Langloan, sulphurous acid gas is passed into the furnace gases in sufficient quantity to combine with the ammonia, which is collected as a mixture of hyposulphite, bisulphite, and sulphate of ammonium. This, when concentrated, is decomposed with lime, in order to recover the ammonia, which is then saturated with sulphuric acid to form sulphate. The sulphurous acid used is obtained by burning nodular pyrites (brasses) with some pyritic ironstone in a peculiar burner, which works with a minimum admission of free air. It is estimated that about 5,500 tons of sulphate of ammonium will be obtained from the existing plant of the Scotch blast-furnaces in the year 1886.

The gases of blast-furnaces carry over with them notable quantities of solid matter in the form of dust, which, accumulating in the flues and gas-mains, requires to be occasionally removed. This principally consists of silica, alumina, ferric oxide, lime, and sulphate of calcium.

The use of dust-catchers is beneficial, as the heating power of the gases is sensibly improved by depriving them of suspended solid matter. When the ores contain zinc, the oxide of that metal accumulates in the throat-flues and under the bell, and must be cleared out from time to time. It also forms considerable deposits in the stoves and boilers. Zinc oxide is especially objectionable when fire-brick stoves are used.

LIFTS OR HOISTS.—When blast-furnaces are situated in the deep valleys of a mountainous country, it not unfrequently happens that all the materials necessary for working them may be delivered by means of a bridge at the top without the aid of machinery. When, however, the country is flat, it becomes necessary to have recourse to mechanical lifts for raising the charges.

In the older iron-works, when erected on level ground, inclined planes are often employed for this purpose, and are usually made with a double line of railway carried on trestle-work, or with a single line and crossings for the return trucks. The inclination given to these does not generally exceed 25° or 30°, and the motive power employed is usually a steam-engine, giving motion to a winding-drum. The truck in most cases consists of a triangular framework with two pairs of wheels, of unequal diameters, supporting a platform on which are placed the iron wheelbarrows used in charging. Chains or wire-ropes are used for raising the load. Where large quantities of material have to be elevated to a considerable height, it is now more usual to employ a perpendicular lift, by which the charges are raised by means of cages moving between vertical guides.

*Lift at Newport.*—The mode of arrangement and the nature of the power employed vary in different establishments, but the woodcut, fig. 72, page 242, copied from a paper read by Sir B. Samuelson before the Institution of Civil Engineers, represents an elevation of the furnace-hoist used at the Newport Iron-Works, near Middlesborough.

The entire lift to the charging platform of the furnaces is 92 feet, and the motive power, instead of being below, as is frequently the case,

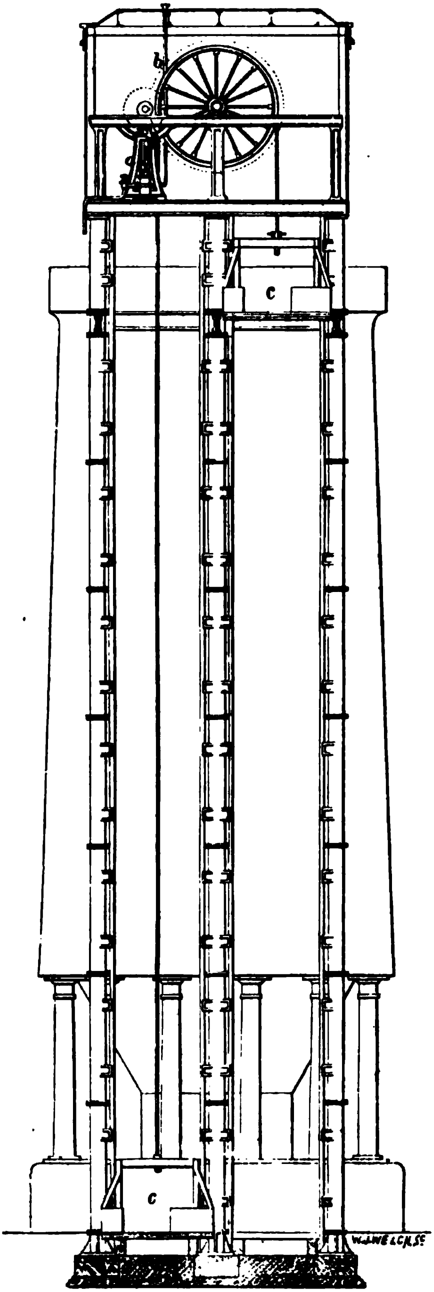


Fig. 72.—Furnace-Hoist, Newport; elevation.

is placed overhead, and consists of a double-cylinder engine, *a*, provided with link motions. The diameter of the cylinders is 8 inches, and the length of stroke 12 inches. On the crank-shaft are two pinions working into wheels on an intermediate shaft. On the middle of the latter is keyed a larger pinion, gearing into the main spur wheel, *b* (represented by a dotted line), 12 feet in diameter, which is flanked on either side by a deeply grooved pulley carrying a steel rope, 1¼ inch in diameter. These ropes fit the grooves with a considerable degree of exactitude, and only pass half round their respective pulleys, the ends being attached, in pairs, to the two cages, *c*. By this arrangement, while one of the cages is ascending, the other is going down, the work being accomplished by the friction of the ropes in their respective grooves.

In order to secure equal tension on both ropes, their attachment to the cage is effected by a double lever, which immediately yields to any unequal stretching of the ropes. The cages are steadied in their upward and downward course by guides fastened to the columns which support the platform. The weight raised at each journey is about two tons, although a much heavier load can be lifted without any slipping of the ropes.

It will be observed that the moment the descending cage touches the ground the strain on the ropes is relieved, so that they will no longer hold sufficiently in the grooves to enable the ascending cage to be raised any higher; this slipping of the ropes renders over-winding impossible. The great length of steam-pipe required for working the engines at this elevation is not found, practically, to be objectionable. The engines usually make about 150 strokes per minute, and, calculating for loading and unloading the cages, they are capable of making one lift per minute, or of raising 120 tons of material per hour.

*Water-Balance.*—The old water-balance lift consists of two cages moving vertically, and guided in the usual way, united by a rope or chain passing over a pulley. Below the floor of each cage is a water-tight box, provided at bottom with a discharge-valve. When the cage with the empty box is at the top of its course, water is run into it until its weight becomes sufficient to overbalance the other cage with its load, the speed of its descent being regulated by a brake on the pulley around which the rope or chain passes. As soon as the descending cage reaches the ground, a projecting spindle in the discharge-valve is forced upwards and allows the water to escape, leaving the cage ready for another ascent as soon as it is loaded. The principle objection to this arrangement is the difficulty of preventing leakage from the tanks, by which the lift-house is constantly kept in an untidy state.

Lifts are also sometimes constructed upon the system of Sir W. Armstrong, where the cage is lifted by the action of a hydraulic ram, of which the course is multiplied by a chain passing over a system of compound pulleys. Pneumatic lifts are extremely convenient, and are much used in the iron-works of this country.

*Furnace-Hoist, Ayresome.*—Figs. 73 and 74 represent a furnace-hoist on the pneumatic principle erected by Mr. Gjers at the Ayresome Iron-Works, Middlesborough, which consists of a 36-inch cylinder, *a*, the whole height of the furnace, made up of flanged cast-iron tubes, lipped, bolted together, and accurately bored throughout. In this cylinder works a heavy piston, lightly packed with a cotton jacket, which also forms a balance-weight, and is sufficiently heavy to balance the table, with four empty barrows, and a portion of the load. From this piston four wire ropes pass over four pulleys, *b*, overhead, down to each

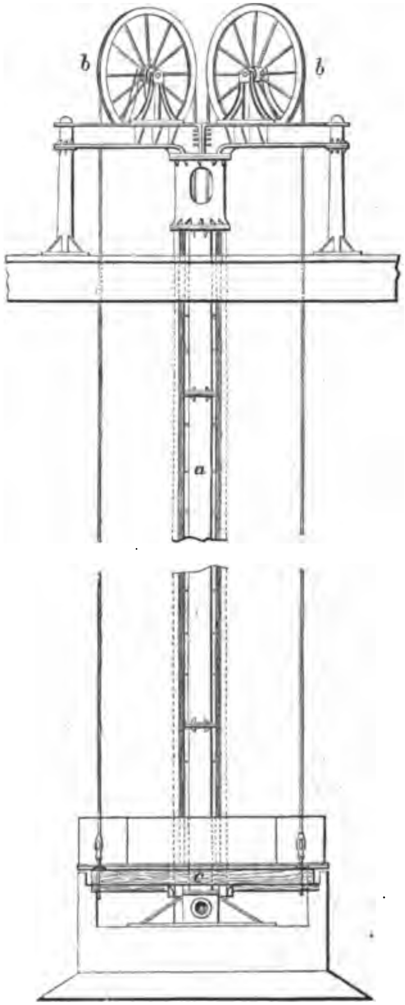


Fig. 73.—Furnace-Hoist, Ayresome Iron-Works; elevation.

corner of the table, which is 15 feet square, surrounds the cylinder, and is guided by four shoes on the table, working in wooden guides on the cylinder. This leaves plenty of room for four barrows being placed round the cylinder, the table having a palisading around with openings on two opposite sides, so that the barrows are run on at one side at the bottom, and run off on the opposite side at the top. The bottom of the cylinder is connected with a pair of single-acting air-pumps worked by a pair of steam-cylinders at an angle of 45°. Between the pumps and cylinder is a reversing-slide, so arranged that, by moving it in one direction, the delivery of air is put in connection with the cylinder, and the exhaust with the atmosphere; by moving it the reverse way, the exhaust is in connection with the cylinder and the delivery with the atmosphere. The table and empty barrows being at the top, the piston will be at the bottom; the engine being started, air is forced into the cylinder under the piston, and about 2 lbs. pressure will lift the piston and bring the table down. When the engine is stopped, the full barrow having been put on the table, the slide is reversed; the piston being now at the top and the engine started, air is removed from under the piston and a partial vacuum produced; the atmospheric pressure acting on the top of the piston now brings it down and the table up, till, arriving at the top, the engine is again stopped. Heavy ironstone barrows, carrying about 50 cwts. of stone, leaving about 40 cwts. unbalanced, require a vacuum of about 4 lbs. to bring the piston down; whereas coke weighing only 20 cwts., with about 10 cwts. unbalanced, is brought up with a vacuum of about 1 lb. The engine is worked, stopped, and started like a winding-engine, the speed at which the table is brought up or down depending upon the speed at which the engine is run.

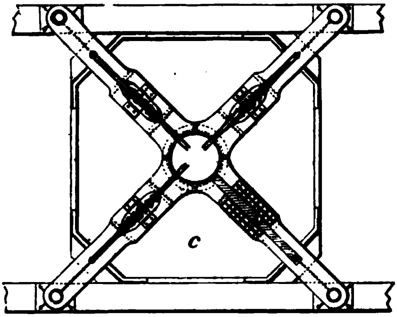


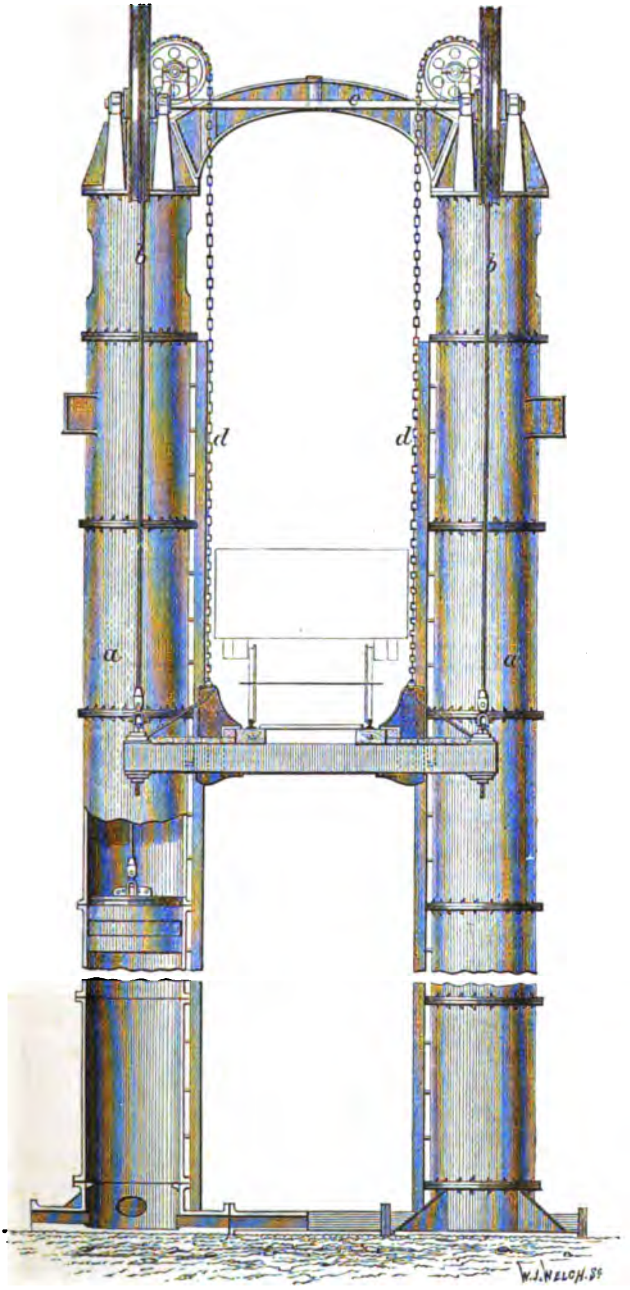
Fig. 74.—Furnace-Hoist, Ayresome Iron-Works; plan.

The cylinder being open at the top, the rope, shackles, and piston-packing are always accessible. The air-pumps are simply a pair of small single-acting blowing-engines, exhausting air by one pipe and delivering it by another, the suction-flaps being on one side and the delivery-flaps on the other.

It will be noticed that the table being connected with the piston by four ropes, no serious accident could happen unless all were to break at the same time. No run-away is likely to take place, as the piston can neither go up nor down faster than air is introduced or removed from below it. The pull on the ropes being through the elastic medium of the air, much less strain is thrown upon them than is the case with direct winding.

*Kiln-Hoist, Ayresome.*—Figs. 75 and 76 are front and side elevations of a hoist employed at the same works for lifting large railway trucks upon the depots and kilns; this is also a pneumatic lift, working exactly on the same principle as that just described, but is arranged with two cylinders, *a*, each of 48 inches diameter, with the table between them. From each piston wire ropes, *b*, pass over pulleys overhead, lap once round, and down to the table corners. The opposite pulleys are keyed on shafts, *c*, as shown, so as to maintain parallelism. There is also a safety chain, *d*, on each side, which ordinarily does not take any weight, but comes into play in case of the breakage of a rope. The weight to be lifted is from 15 to 16 tons, and it requires a vacuum of about 6 lbs. to bring the pistons down and the table up, the balance being such that it requires a pressure of 4 lbs. to bring the empty table down. The engine to work this lift is precisely similar to that for the furnace-lift, but is worked at a much slower speed. The height to which the trucks are lifted is 35 feet, and with four furnaces it will be required to lift at least 6,000 tons per week, including the weight of the trucks.

An empty-waggon drop is also employed at the Ayresome Works, which acts on the pneumatic principle, and is constructed similarly to the lift, with this difference, that the cylinders are only 36 inches in diameter, and the ropes, table, &c., lighter. Instead of the lift-pistons having a space beneath them, and being connected with an engine, a bottom is put into each cylinder on which the pistons come down. Close above this bottom, on each cylinder are valves, one set communicating with the atmosphere, and the other through a pipe with the general blast-main, both sets being actuated by handles on the top of the depôts. The table being at the top and the pistons at the bottom, and both sets of valves being closed, the empty truck is run on; the balance of the pistons being such that the table, with truck on, is somewhat heaviest. As long as the valves at the bottom are shut, however, the table cannot raise the pistons, as immediately they move a partial vacuum is formed below them; the moment, however, the valve communicating with the atmosphere is opened, the extra weight of the table pulls the pistons up, drawing air after them through the valves. This constitutes an air-brake, and is entirely under control; the descent can be stopped at any time by shutting the valves, and the speed of the fall regulated at pleasure. On the table arriving at the bottom the truck is run off, but before doing so, the attendant changes the handles at the top—that is, shuts the valves to the atmosphere, and opens the set communicating with the



. 75.—**Kiln-Hoist, Ayresome Iron-Works; front elevation, partly section.**

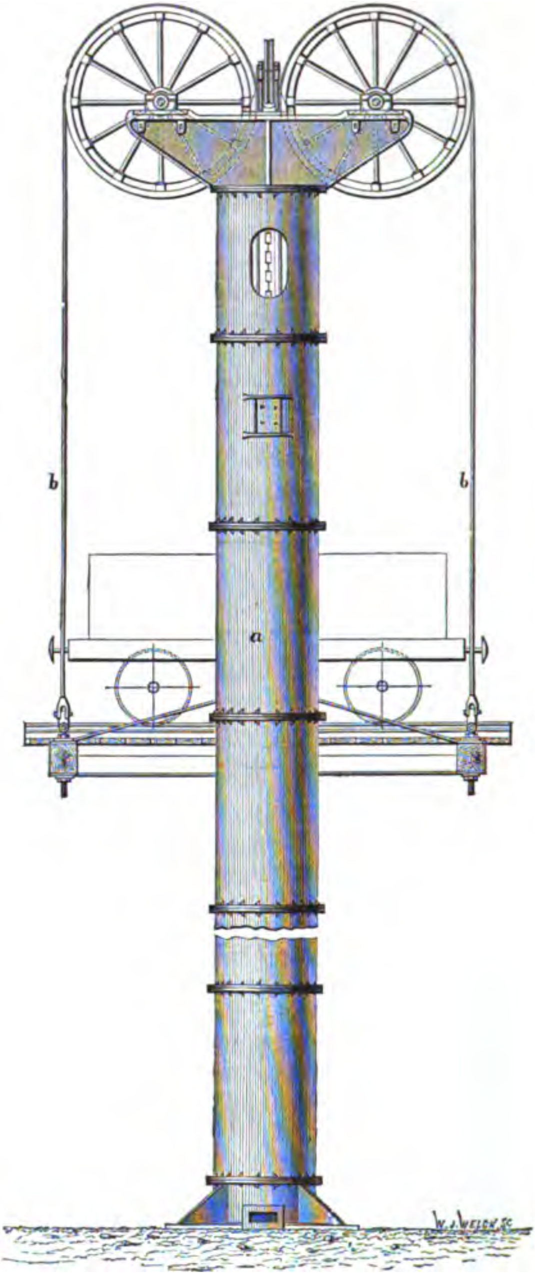


Fig. 76.—Kiln-Hoist, Ayresome Iron-Works; side elevation.

blast-tubing, thus producing a pressure under the piston of 4 lbs. This is for the purpose of supporting the heavy pistons when the truck is run off; but it also has an ulterior object, as the balanced weight of the pistons is such that they will descend against the 4 lbs. pressure, thus forcing the two cylindersful of air into the blast-tubing. It will thus be seen that for every empty truck which descends this drop a certain amount of air is forced into the blast-main. Direct-acting steam-hoists are also frequently employed in the North of England for lifting trucks to the depôts or kilns.

## SMELTING.

*Fuel used in the Blast-Furnace.*—The fuel used in the blast-furnace is usually either charcoal or coke, but both raw coal and anthracite are likewise extensively employed. Owing to its freedom from sulphur and other impurities, charcoal yields pig-iron of superior quality to that smelted from the same ores with coke, but the immense consumption of wood entailed, renders the manufacture of charcoal iron difficult in any but thickly wooded districts, or such as have cheap communication with similar districts in the neighbourhood. In England charcoal furnaces are entirely abandoned, and their number is constantly diminishing on the Continent of Europe. In the Styrian Alps the local supply of charcoal is supplemented by importations from the Lower Danube, involving a railway journey of several hundred miles. Peat and dried wood have occasionally been used in admixture with charcoal, but the experiment has never proved to be permanently satisfactory. In the United States the production of charcoal-pig is still comparatively large, though less than it was formerly.

Coke for blast-furnace use is valued in proportion to its freedom from ash and sulphur. The former may range from 5 to 15 per cent., and the latter from 0·3 to 1·0 per cent. Its physical structure is also of importance; the hard silvery columnar varieties, produced at the highest heat, being preferred to the dark, dull kinds produced at lower temperatures. Formerly it was considered that the coke from whole coal was stronger than that made from slack, but opinion has changed upon this point, and it is now common to grind caking coal as a preliminary to coking. It is still matter of opinion whether extremely dense coke is the most economical. What is most required is that it should resist oxidation by heated carbonic acid in the upper regions of the furnace (to avoid an unnecessary production of carbonic oxide), but burn freely when brought under the influence of the blast from the tuyers. A hard and at the same time somewhat coarsely vesicular fuel seems to be the best. Sir I. L. Bell has found the smelting value of coke made in ovens utilizing the bye-products of the gases to be about 10 per cent. below that made in the ordinary beehive oven. Anthracite, although one of the purest of mineral fuels, is from its great density and impermeability less advantageous to the iron-smelter than might be expected. In South Wales a further disadvantage is experienced from the tendency to decrepitation, which makes it necessary to work with an open tymp, so as to be able to clear out the small particles of anthracite that have escaped combustion and accumulated in the hearth. The anthracite of Pennsylvania does not decrepitate to the same extent when heated, and is therefore better suited for blast-furnaces than that of South Wales. It is much used in the North-Eastern States, especially in Pennsylvania and New York, but even there it is being largely replaced by coke.

Raw coal for blast-furnaces must be of a non-caking kind, and when of a suitable quality it is perhaps the most advantageous fuel. The principal localities in which it is used are the West of Scotland and North Staffordshire, South Staffordshire, and Derbyshire. It is used to some extent in admixture with coke. In the United States the so-called block coal, which resembles the splint coal of Scotland, is used in the blast-furnaces of Indiana.

*Blowing-in.*—The blowing-in of a blast-furnace is an operation necessitating considerable care, since if too hastily effected great injury to the masonry would result. When a furnace has been made ready for blowing-in by building up the tuyer-holes, &c., a quantity of rough dry timber is placed in the hearth, filling it to the height of from 5 to 6 feet; on this is piled coke until it reaches and fills the boshes. Fire is now applied to the timber, which quickly communicates it to the coke above, and regular charges of calcined ironstone, limestone, and coke are added until the materials reach the throat of the furnace. The relative amounts of ironstone, limestone, and coke employed vary in different localities; but in the neighbourhood of Merthyr Tydvil, in South Wales, they are, according to Truran, often in the proportions of 5½ cwts. calcined ironstone, 1¾ cwt. limestone, and 4 cwts. of coke.

The furnace having been in this way gradually filled to the throat, the blast is turned on to the extent of about one-fifth of the volume usually employed. For a furnace intended to be blown with 4-inch nozzles, the first set should have a diameter of 1¾ inch; after blowing about thirty hours, these may be replaced by others 2½ inches in diameter; and at the expiration of three days these may be exchanged for 3¼-inch pipes. By the close of the third week the size may be increased to 3⅜ inches, and in four or five weeks after blowing-in, full-sized pipes may be used.

Shortly after the admission of the blast, the workmen commence clearing the hearth below the tuyers for the reception of slags, which begin to make their appearance about twelve hours after beginning to blow. In twenty-four hours these will have filled the bottom of the hearth.

The metal, which usually makes its appearance about twelve hours after the cinders, will, in a furnace of a capacity of 7,500 cubic feet, accumulate to the amount of from 3 to 4 tons at the expiration of sixty hours after the admission of the blast. Eighteen hours later, another casting of 2 tons of metal may be made, and thenceforward the castings may be performed at the usual fixed periods. The old method of supporting the charges of a furnace on a "scaffold" during the process of blowing-in is now seldom resorted to. When this system is adopted, a grate of iron bars is made across the hearth on a level with the top of the dam-plate, and as soon as the first charge of ironstone reaches this point, the bars are withdrawn and the blast turned on.

*Descent of Charges.*—The respective amounts of ironstone, flux, and fuel required for the production of iron of a given quality having been determined, it is important that their relative proportions should be maintained unaltered in the consecutive charges by the workmen engaged in filling, and, for their guidance in this respect, weighing-machines are placed in the various barrow-roads over which the charges are transported. In small charcoal furnaces, the different ores and fluxes are usually made up in suitable proportions in alternate layers one above the other, and the mixture obtained by making a vertical section of the heap is charged into the furnace. In coke furnaces, the ore and limestone are charged separately, or they may be placed in the charging-barrows in distinct strata.

In order to obtain regularity in the action of a furnace, the column of descending materials should be uniformly heated by the ascending current of hot gases. There is, however, in practice, considerable difficulty in attaining this result, since the upward gaseous current follows the sides of the furnace, whereas the flow in the centre of the mass is comparatively insignificant. The descent of the solid charges, on the other hand, takes place under very different conditions, because the fragments of which they are composed are sensibly retarded by friction against the masonry of the furnace. The central portions are not thus affected, and, descending more rapidly, they have a tendency to become less heated than portions of the mixture occupying a position in which the motion is less gradual, and where a more elevated temperature prevails.

From this cause the central portions of the successive charges, deposited in the throat of the furnace in approximately parallel layers, overtake the sides of those which precede them in the series, and thus, at a certain depth below the mouth, the contents of the furnace become intimately mixed, a nearly uniform heat being the result.

The distribution of the materials in a furnace is also materially affected by the arrangements adopted for charging, which may leave the upper surface of the column either horizontal, as an upright, or an inverted cone, or as a combination of the two, resulting in a conical heap with a funnel-shaped depression in its centre. When the upper surface of the charge assumes the form of a cone, the fragments of ore and fuel dropped upon it from above take up positions in conformity with their differences of form and density. A large proportion of the ironstone will remain where it first falls, while the lighter fuel, which is usually in larger fragments, rolls down the slope and arranges itself around the periphery of the base, thus establishing the worst possible combination of circumstances by allowing a free passage to the gases around the sides, while a dense core of almost impermeable ore accumulates in the centre.

When the charges are distributed around the circumference of the throat, the surface forms a conical cup, the lighter fragments rolling inwards towards the centre, while the ore accumulates in the vicinity of the walls. This tendency of the fuel and larger masses of ore to settle in the middle, forming a central, readily permeable column, results in a more equal distribution of the draught over the entire horizontal section, while the bulk of the ore descends slowly through the region most highly heated by the current of ascending gases. These conditions are favourable to uniform and economical working, but in the case of furnaces having wide throats, the central draught may sometimes become so active as to result in an undue consumption of fuel, and the constant contact with ironstone is liable to produce an erosive action on the brickwork of the lining.

*Tapping.*—The removal of the liquid metal is called *tapping*, and is effected by piercing, with a long bar, a plug of sand and clay with which, during the previous operation, a hole communicating with the bottom of the hearth has been closed. Before proceeding to tap, the workmen prepare the moulds or *pig-bed*, fig. 77. This consists of a series of parallel depressions of the shape of the pig required, which are connected by feeders, *b*, with the main channel, *a*, communicating with the taphole. The blast is now shut off, the fore part of the hearth is opened, and the plug of refractory material removed; this allows the melted iron to flow into the lowest line of moulds, and when these are filled the current is stopped by a dam of sand at *d*, which allows the next row to fill, and so on until the hearth is emptied. The pig-beds, as a rule, are made of ordinary siliceous sand, but sometimes granulated slag is preferred. For forge purposes and in making manganiferous metal, heavy chill-moulds of cast-iron are used.

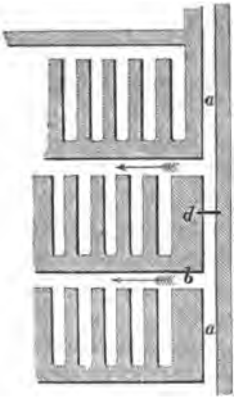


Fig 77.—Pig-Bed.

The semi-cylindrical bars or *pigs* are united by one of the shape of the channels *b*, called a *sow*, from which they are easily separated by being broken off at the point of connection. When the whole of the metal has been drawn off, the blast is again admitted into the furnace, and smelting operations are repeated as above described, until, from the quantity of metal accumulated, a second tapping becomes necessary.

*Blowing-out.*—Whenever it becomes necessary to put a furnace out of blast, for the purpose of repairs or for any other reason, the burden is for some time reduced, in order that the temperature of the hearth may be increased, for the purpose of removing obstructions only fusible at a high heat.

The gas-tubes and other metallic fittings of the throat are next removed, and charging having been suspended, the contents of the furnace are allowed to burn down; the last tapping is made from a point as low down as possible in the hearth.

The hearth is frequently found to be more or less obstructed by imperfectly agglomerated masses of malleable iron; and detached crystals, or even lumps, of considerable size, of a copper-coloured substance, $Ti_5CN_4$, which was formerly supposed to be metallic titanium, are found adhering to the brickwork.

It sometimes happens that, either through want of material, or by some accident to the machinery, it becomes necessary to suspend the operations of a furnace during several successive days. This may often be done by hermetically closing the throat and tuyer-holes with sand or clay; but should the interruption extend over a period of more than a week, cooling takes place to such an extent as to cause agglomerations liable to render the abandonment of the furnace necessary.

VARIETIES OF PIG-IRON.—The metal produced in the blast-furnace is known by the general names of *pig-iron*, *pig-metal*, or *cast-iron;* in French *fonte*, and in German *Roheisen.* Apart from the more manganiferous varieties, for which special names are adopted, the classification employed is a numerical one, depending upon the greyness of the fracture. The most coarsely crystalline and darkest, or most graphitic metal, is rated as No. I., while No. II. is of a smaller crystalline texture, and so on, the fracture becoming finer-grained and more uniform up to Nos. IV. or V., at which point the limit of mottled-iron is reached. The fracture showing alternations of grey patches in a white base or the reverse, gives rise to qualities known as *strong-* and *weak-mottled* respectively. When the granular grey patches are entirely absent, the fracture becomes nearly uniform, with a dull silvery lustre, or that of *white-iron.*

The difference between white and grey pig-metal depends chiefly upon differences in the amount of graphitic carbon present, which is almost entirely absent in the former and reaches a maximum in the latter. The separation of graphite is, however, promoted by the presence of silicon, and therefore the higher or "richer" classes of grey-metal, Nos. I and II., usually contain a notable proportion of that element. These are especially valuable for mixing purposes in foundries, although not fitted for the production of castings alone. The standard number for general foundry work is No. III., and it is this quality which it is sought to produce on a large scale in Cleveland, the higher numbers requiring a larger consumption of coke.

The special qualities known as Nos. I. and II. Bessemer pig differ from the same Nos. of foundry iron by their freedom from phosphorus. They are smelted from magnetite, hæmatite, or other of the purer kinds of ore, and may contain a large amount of silicon with advantage.

Grey pig-irons low in silicon, such as are smelted from ores containing a small proportion of manganese with charcoal, or, as is still done to a small extent, with coke and cold-blast, are known as strong foundry irons. These are specially adapted for the production of objects with hardened or chilled surfaces by casting in metal or chilled moulds, such as rolls for forges, paper-, flour-mills, and railway wheels. The car-wheel iron of America and the Staffordshire cold-blast pigs are examples of this kind of metal, which is also applied in the production of malleable castings and many small objects of cast-iron which are superficially converted into wrought-iron by heating them in powdered hæmatite. For this purpose almost entire freedom from manganese is necessary. The type of this class of metal is the Lorne pig-iron, formerly smelted with charcoal from Cumberland hæmatite.

White pig-metal being principally used for conversion into malleable iron, should contain a minimum of carbon, and as little silicon, phosphorus, and sulphur as possible. These conditions, however, can only be realized in smelting very pure ores, such as the spathic ores of Styria or the Swedish magnetites, with charcoal, when the metal obtained is almost entirely free from silicon and sulphur. When mineral fuel is used, it is difficult to prevent the metal taking up sulphur unless the ore contains manganese, as the use of sufficient limestone flux for complete desulphurization is excluded by the necessity of producing a moderately fusible slag, otherwise the metal will take up silicon and become grey. The necessary fusibility in such cases is contributed by other bases, such as the oxides of iron and manganese, producing a dense, dark-coloured slag, which is excessively corrosive or scouring in its action on the furnace. White-iron is essentially the product of a heavily burdened furnace working with rich and easily fusible materials, such as hæmatite and forge-cinders, and consuming a minimum of fuel per ton of metal produced. It was at one time largely made in South Wales for rail-making, but since the use of steel for this purpose has become general, its importance has considerably diminished.

When white-iron is produced by furnaces that are burdened for grey-metal, it indicates some derangement in the working conditions, which must be corrected by modifying the fluxes and increasing the relative proportions of fuel to burden.

*Manganiferous Pig-Metal.*—The presence of manganese in pig-iron is advantageous, except in the case of metal intended for making malleable castings, or when a high proportion of silicon is desired. The reduction of manganese oxide can only be effected by contact with solid carbon at a temperature considerably above that at which manganese silicate is formed by the contact of manganous oxide and silica, and this silicate, when once formed, can scarcely be reduced. The loss of a considerable portion of the metal in the slags is therefore a necessity in smelting manganiferous ores. Manganese takes up twice as much carbon as iron, but retains it in the combined condition up to about 6 per cent., giving a white metal with a more or less platy crystalline structure. When the plates are small, so that the fracture has a radiated fibrous or finely columnar structure, the metal is known as *Weissstrahl* in Germany, or *fonte-rubané* in France.[1] This contains up to about 4 per cent. of manganese and combined carbon, and is the principal product of furnaces smelting spathic ores with coke for forge purposes, being easily made, and valuable as a material for conversion into wrought-iron and puddled steel. When the proportion of manganese amounts to 8 or 10 per cent., the metal has a large, platy, crystalline fracture, the separate plates being often more than an inch broad, giving the structure characteristic of *spiegeleisen* (mirror iron), which term, in the contracted form of "*spiegel*," is now in general use in every iron-making country. The characteristic spiegel structure is apparent in metal containing 20 to 25 per cent. of manganese, but with higher proportions of the latter the fracture becomes fine-grained, generally resembling that of a hard white bronze, and the metal is no longer magnetic. With still higher proportions of manganese, 50 per cent and upwards, an acicular crystalline structure is developed, somewhat resembling that of sulphide of antimony. These higher manganese iron alloys, known as *ferro-manganese*, have been made up to a maximum content of about 88 per cent. of manganese and 7 per cent. of carbon. They are exclusively used as additions in the final operations of steel-making by the open-hearth and Bessemer processes.

As manganese oxidizes very readily, the characteristic colour of a manganiferous pig can only be seen on a freshly fractured surface. The surfaces of spiegeleisen are, when well crystallized, generally tarnished with a steel-blue iris, and the richest ferro-manganese oxidizes and disintegrates when exposed to the air for any length of time. From the analyses of different samples of ferro-manganese given on p. 257, it will be seen that the amount of silicon and phosphorus increases with that of manganese.

*Siliciferous Pig-Metal.*—Pig-iron in which the proportion of silicon notably exceeds that of carbon, while retaining the characteristic structure of the higher foundry Nos., is lighter in colour. This so-called "glazed-" or "glazy-iron" is produced when the furnace is in the highest working condition, *i.e.*, when lightly burdened and developing a maximum temperature in the hearth, with highly heated blast. According to M. Athaire, the use of aluminous fluxes as a means of rendering the slags refractory, also favours an excessive reduction of silicon in furnaces burdened for grey-iron. To a very limited extent glazy-iron may be used in foundry mixtures; but as it is too weak to be made into castings alone, and cannot be converted into malleable iron on account of the enormous waste of fettling in the puddling-furnace; it is rarely made except by accident. A tendency to over-reduction of silicon is corrected by lowering the blast temperature and increasing, for a time, the ore burden in the furnace.

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[1] Literally white-rayed and ribboned-metal. There is no English equivalent term, as it is not made, as a rule, in this country.

The richer kinds of manganiferous metal may be made to take up a much larger proportion of silicon than ordinary pig-iron, and alloys of this kind, with 10 per cent. and upwards of silicon, known as *ferro-silicon* [1] or *silicite*, are now made for use in the production of mild steel castings. A small addition made to the metal in the ladle prevents the formation of blow-holes in the finished casting.

*Phosphorus Pig-Metal.*—The method of dephosphorizing in the converter introduced by Thomas and Gilchrist has created a demand for a special variety of pig-iron, containing a maximum of phosphorus and a minimum of silicon; exactly the opposite of what was required in the Bessemer process, as originally carried out. Other essentials are the presence of a notable quantity of manganese and freedom from sulphur. This iron is produced in large quantities from the manganiferous brown iron ores, of Ilsede in Hanover containing phosphorite, with the addition of puddling-furnace cinder. It resembles ordinary white-iron or low spiegeleisen, and contains from 3⅓ to 5½ per cent. of manganese, and from 2 to 3 per cent. of phosphorus. At Seaton-Carew, Durham, similar so-called *Thomas pig-iron* is made with as much as 7 per cent. of phosphorus.

The power of resistance of cast-iron to strains variously applied differs in accordance with the quality and composition of the metal. No. 1 pig is soft and deficient in strength, as compared with lower qualities made from the same ores, and, consequently, for foundry purposes it is customary to so blend the different kinds of metal as to obtain a mixture suitable for the casting to be produced. Silicon is believed to prejudicially affect the strength of cast-iron, and it may be from the presence of a larger amount of this element in hot-blast metal that its strength is inferior to that smelted by the cold-blast.

The maximum and minimum limits of strength of British pig-iron, as deduced from experiments made at the Woolwich Arsenal (1856–59), are as follow:—

| | Minimum. | Maximum. |
|---|---|---|
| Specific gravity . . . . . | 6·886 | 7·289 |
| Tensile strength per square inch . | 4·85 tons | 14·05 tons. |
| Transverse „ „ „ . | 1·37 „ | 4·47 „ |
| Torsional „ „ „ . | 1·74 „ | 3·44 „ |
| Crushing „ „ „ . | 22·54 „ | 58·42 „ |

Tensile strength was determined by tearing asunder short columns of 1·3 inch in smallest diameter. Transverse strength was determined by taking the mean of a number of values derived from breaking bars 22 inches long and 2 inches square. Torsional strength was determined on round bars 8 inches long between the points at which the twisting force was applied. The crushing strains were deduced from cylinders 1·3 inch high and 0·6 inch in diameter.

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[1] *Silico-ferro-manganese* would be a more exact description.

The lowest values were obtained from iron made from sandy brown ores, and the best from hæmatite and from argillaceous carbonates, smelted either together or separately with cold-blast.

COMPOSITION OF PIG-IRONS.

| | C.[1] | | | Si. | P. | S. | Mn. | Fe. |
|---|---|---|---|---|---|---|---|---|
| | a. | b. | c. | | | | | |
| *Grey-Irons.* | | | | | | | | |
| 1. Coltness, No. I. Foundry | 3·30 | 0·20 | ... | 3·50 | 0·98 | 0·02 | 1·58 | ... |
| 2. Langloan, No. II. „ | ... | ... | 3·22 | 2·34 | 0·51 | ... | 1·21 | ... |
| 3. Clarence, No. III. „ | 2·16 | 0·65 | ... | 1·79 | 1·80 | 0·10 | 0·56 | ... |
| 4. Workington, No. I. Bessemer | 3·85 | 0·24 | ... | 2·20 | 0·10 | 0·05 | 1·15 | ... |
| 5. Hörde Manganiferous „ | 3·20 | 0·35 | ... | 2·20 | 0·30 | 0·08 | 6·70 | ... |
| *Strong Foundry Irons.* | | | | | | | | |
| 6. Lerbach, Charcoal Cold-blast, No. II. | 3·85 | 0·48 | ... | 0·79 | 1·22 | 0·02 | ... | ... |
| 7. Bowling, Cold-blast, No. I. | ... | ... | 2·99 | 0·97 | 0·50 | 0·05 | ... | ... |
| 8. Grazebrooks, Cold-blast, No. I. | 3·07 | ... | ... | 1·48 | 0 43 | 0·03 | 0·96 | ... |
| 9. Finspong, Gun-foundry Iron | 2·17 | 1·75 | ... | 0·97 | 0·05 | 0·12 | 0·19 | ... |
| *Forge Pig-Irons.* | | | | | | | | |
| 10. Cleveland, mottled | ... | ... | 3·55 | 0·66 | 1·05 | 0·35 | 0·79 | 93·95 |
| 11. Dowlais „ | ... | ... | 2·95 | 1·96 | 0·63 | 0·28 | 0 23 | ... |
| 12. Cleveland, white | ... | ... | 3·20 | 0·64 | 1·32 | 0·20 | 0·60 | ... |
| 13. Dowlais „ | ... | ... | 2·84 | 1·21 | 0·64 | 0·46 | 0·14 | ... |
| 14. Luxemburg „ | 1·10 | 0·47 | ... | 0·40 | 1·62 | 0·30 | 0·44 | ... |
| 15. Lorne, Charcoal, white | ... | ... | 3·69 | 0·28 | ... | 0·01 | 0·06 | ... |
| *Silicon and Phosphorus Pig-Irons.* | | | | | | | | |
| 16. Glazy-Iron, Clarence | 2·59 | 0·79 | ... | 5·13 | 1·12 | 0·17 | 0·77 | 88·18 |
| 17. St. Louis, Silicon-Pig | 3·44 | 0·21 | ... | 4·89 | 0·06 | ... | 0·84 | ... |
| 18. Ilsede, Basic-Pig | ... | ... | 2·68 | 0·11 | 3·29 | 0·04 | 3·84 | ... |
| *Manganiferous Pig-Metal.* | | | | | | | | |
| 19. Eisenerz Weissstahl | ... | ... | 3·40 | 0·09 | 0·02 | 0·77 | ... | ... |
| 20. Wissen Spiegeleisen | ... | 4·10 | ... | 0·25 | 0·03 | 0·06 | 11·10 | ... |
| 21. French „ | ... | ... | 5·25 | 0·46 | 0·08 | ... | 20·50 | 75·86 |
| 22. „ Ferro-manganese | ... | ... | 6·00 | 0·46 | 0·13 | ... | 51·80 | 41·70 |
| 23. „ „ | ... | ... | 5·40 | 2·73 | 0·24 | ... | 74·00 | 18·20 |
| 24. „ „ | ... | ... | 6·50 | 1·49 | 0·29 | ... | 87·90 | 6·78 |
| 25. „ Ferro-silicon | ... | ... | 2·65 | 10·20 | 0·18 | ... | 20·50 | 66·75 |

PRODUCTION OF BLAST-FURNACES.—The greatly increased production of the modern blast-furnace, as compared with those of older date, is partly due to its larger size, and partly also to the proportionately large amount of blast with which it is now supplied. The time necessary for the complete reduction of the ore, previously to actual fusion, is dependent on many variable elements, such as its richness, composition, porosity, and density, the nature of the fuel, &c. It is therefore necessary, in order to obtain any given result, with regard to either quality or produce, to ascertain by actual experiment, for each particular furnace, the amount of blast, burden of ore, and admixture of fluxes which should be employed. The more completely the materials are exposed to the action of the ascending gaseous current, all other conditions being the same, the shorter will be the time necessary for reduction, and it is consequently important that, by the use of suitable charging and gas-collecting apparatus, the flow of gases through the mass should be rendered, as far as possible, uniform. The free escape of gases from the top of the furnace must specially be provided for, and on this account methods based upon their collection above the surface of the charges are to be preferred to those in which lateral flues penetrating the walls below the level of the throat are made use of. If the pressure of the blast delivered to a furnace be kept constant, while the volume is increased, a tendency to produce white-iron will be developed. On the other hand, by increasing both pressure and temperature, especially if the ores be of a refractory character, the production of mottled or grey-iron is likely to be the result.

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[1] Of the three columns under C., the first, *a*, gives the amount of graphitic carbon, the second, *b*, that of combined carbon, and the third, *c*, the total amount of that element present.

*Charcoal Furnaces.*—The Styrian furnaces, which are remarkable for their large production as compared with their small volume, consist of two truncated cones united by their bases; the throat is very narrow, and they differ from other blast-furnaces in being without a fore-hearth. The metal and slag are allowed to accumulate in the hearth, whence they are removed by frequent tappings, as many as sixteen casts being sometimes made in the course of twenty-four hours. The ores treated are chiefly spathic carbonates, poor in manganese, and more or less changed by oxidation and loss of carbonic anhydride into brown hæmatite. In order to free the ironstone as completely as possible from sulphur, it is, after roasting, exposed to the action of the weather for a period of from two to three years. The ores treated contain from 35 to 55 per cent. of metal, and the object sought is the production of white-pig for the manufacture of bar-iron with a minimum expenditure of charcoal. In order to attain this end the furnaces are worked with a very heavy burden, care being taken to counteract the tendency to form obstructions by introducing at regular intervals charges of fuel without ore. Some of the ores treated are so constituted as not to require the addition of fluxes, but they generally contain so large a proportion of lime as to render a mixture of siliceous and argillaceous materials necessary.

Von Fischer's furnace, Vordernberg, is one of the smallest in the world, its total height being 28 feet, its width at the boshes 6 feet, and its cubic capacity 452 feet. The usual charge of this furnace consists of 223 lbs. roasted ore, 15 lbs. clay, and 4 lbs. of the granulated pig-iron recovered by stamping and washing the slags; to this are added 95 lbs. charcoal. This furnace is tapped at intervals of ninety minutes, fourteen charges, including a blank one of fuel without burden, being made during the same period; the daily production of pig-iron is 7½ tons.

Von Fridau's furnace, in the same district, is somewhat larger than the above; its height is 42 feet, its diameter at the boshes is 7½ feet, and its capacity 1,052 cubic feet. The full charge consists of 6 cwts. of roasted ore, to which are added 10 per cent. of clay, and about 10 lbs. of granulated metal from the slags. Fifteen and a half cubic feet of soft pine-charcoal, weighing 101 lbs., are employed with each charge, and the burden of ore is gradually raised from 3 to 6 cwts. per charge, and afterwards successively diminished in a similar way; a blank charge of fuel without ore being introduced at each change from an increasing to a decreasing burden. The production of this furnace varies from 18½ to 20 tons daily, and the tuyers, which incline at an angle of 5°, cause the hearth, to some extent, to act as a refinery.

The largest Styrian charcoal furnace at Trofaiach, 52 feet high, 9½ feet diameter of boshes, and of about 2,050 cubic feet capacity, smelts from 90 to 125 charges daily, producing from 30 to 40 tons of metal. The spathic ore, 46 per cent. produce, is used in the proportion of ⅘ roasted to ⅕ raw, together with siliceous clay or puddling-furnace cinder as flux. When the latter is in more than fluxing proportions, a corresponding addition of caustic lime is added. The consumption of charcoal is 13½ cwts. per ton of white forge-iron, with blast at about 350° C.

The blast-furnaces employed in Sweden are, in many respects, similar to those of Styria, but they are provided with a small and narrow fore-hearth. Their capacity is usually inconsiderable, varying from 600 to 2,500 cubic feet. The best varieties of Swedish ores are known as "self-fluxing," and contain earthy materials in such proportions as to afford fusible slags without further addition. The ores of Dannemora, Langbanshytta, and Langshytta are of this description, and contain from 50 to 60 per cent. of iron. At the last-named locality the charges, even after the addition of from 3 to 5 per cent. of limestone, sometimes contain above 60 per cent. of iron. The more siliceous hæmatites and micaceous ores are mixed with calcareous magnetites, and fluxed with dolomitic limestone; the average amount of iron in the charges varies from 35 to 50 per cent.

The siliceous itabirite of Nora requires an admixture of 25 per cent. of limestone, and at Taberg, where the ore smelted consists of magnetite disseminated in an eruptive greenstone, the charges contain only about 20 per cent. of iron. At Dannemora, the blast is heated to from 80° to 100° C., and throughout Sweden the temperature does not usually exceed 200° C. The waste gases are withdrawn through an aperture in the side of the furnace 12 to 15 feet below the throat, and are employed both for heating the blast and for the calcination of ores. Cold-blast is used at Finspong for the production of gun-foundry iron.

The average weekly production of the Swedish furnaces ranges from 30 to 60 tons of pig-iron, but at Langshytta the weekly make is 117 tons, at Sandviken 104 tons, and at Langbanshytta 75 tons per furnace. In 1882 the average number of days that a furnace was in blast in Sweden was 230, and the make 2,112 tons, or about 9 tons daily.

The consumption of charcoal varies from 16 to 17 cwts. per ton for white and mottled pig-iron produced, and from 21 to 22 cwts. per ton for grey-metal suitable for foundry purposes, or the preparation of Bessemer steel; the poor magnetic ores of Taberg require as much as 50 to 60 cwts. of charcoal to produce a ton of pig-iron.

At the well known iron-works at Finspong, Ostgothland, the pig-iron intended for making cannon is run directly from the furnace into the moulds; whereas in other foundries it is usual to re-melt the pig-iron in reverberatory furnaces. The charge in 1857 was, according to Tunner, composed as follows:—

| | Lispunds.[1] | Lispunds. | Lbs. Avoirdupois. |
|---|---|---|---|
| Ferola ores | 29·8 | 42·0 | 625·80 |
| Jerna „ | 5·4 | | |
| Petäng „ | 4·1 | | |
| Stenbo „ | 2·7 | | |
| Scrap cast-iron | ... | ... | 22·35 |
| Iron-borings | ... | ... | 22·35 |
| Limestone | ... | ... | 96·85 |
| | | | 767·35 |
| Charcoal | ... | Tunnas.<br>9 | Imperial bushels.<br>36 |

The Ferola ores chiefly consist of granular magnetite and quartz, with a little oligoclase, hornblende, and iron pyrites; the Jerna ore is a richer and less compact magnetite, associated with the same minerals as those from Ferola. The Petäng ores are similar, but are more finely granular, and contain a considerable percentage of manganese; the Stenbo ore is a mixture of magnetite and spathic carbonate of iron. These ores yield from 48 to 52 per cent. of pig-iron of great strength, but contain a notable quantity of sulphur; this, instead of impairing the quality of the metal, is believed to increase its strength.[2]

At Marquette, Lake Superior, the ores smelted are a brown hæmatite containing, on an average, 40 per cent. of iron, and a specular schist yielding 60 per cent. of that metal. These ores are mixed in such proportions as to yield 55 per cent. of pig-iron, and are treated in a furnace 40 feet in height, 11 feet in the boshes, and 4 feet at the throat; the gases are collected in an annular flue enclosed by an iron cylinder. The blast is introduced at a temperature of 330° C., and at a pressure of about 2 lbs. per square inch, through two tuyers, each $3\frac{1}{2}$ inches in diameter, on opposite sides of the hearth. A crystalline limestone is used as flux, to the amount of about 10 per cent., and the consumption of charcoal is approximately 25 cwts. per ton of pig-iron. The weekly production is from 125 to 130 tons of fine-grained dark-grey pig, suitable either for foundry work or for the manufacture of Bessemer steel.

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[1] Each of 14·9 lbs. avoirdupois.

[2] With respect to the influence of sulphur on the quality of iron, Dr. Percy remarks: "I have particularly interrogated the intelligent managers of iron-works, from every part of England, as to their opinion concerning the influence of sulphur, in certain proportions, even on bar-iron, and they have, without exception, expressed the opinion that it is not unfavourable to strength, however it may interfere with the finish on the surface of the metal."—*Iron and Steel*, p. 554.

The newer American charcoal furnaces are from 50 to 60 feet high, 9 to 12 feet in the boshes, and from 2,400 to 3,000 feet cubic capacity, with very steep boshes (from 73° to 81°). The make is from 300 to 350 tons weekly. The Martel furnace at St. Ignace, Michigan, 52 feet high, with 10 ft. 6 in. boshes, 7 feet throat, $5\frac{3}{4}$ feet hearth, and 2,800 cubic feet capacity, has made up to 420 tons per week, smelting specular hæmatite of 60 per cent. produce with $13\frac{1}{2}$ cwts. of charcoal per ton. The ores are very pure, requiring only 4 cwts. of dolomitic flux, which produces only about 5 cwts. of cinder per ton. The blast is heated by two Whitwell stoves, 59 × $14\frac{3}{4}$ feet, to 800° C. In 1884 seventy-eight charcoal furnaces were blowing in the United States, and made 408,400 tons of pig-iron, with a consumption of 452,600 tons of charcoal. In Michigan 153,500 tons of pig-iron were made with 132,800 tons of charcoal.

*Coke Furnaces.*—The ores smelted in the Siegen district are principally spathic carbonates and brown hæmatites; the former contain a considerable quantity of manganese. The spathose ores are, for the most part, obtained in the immediate neighbourhood, but a considerable portion of the hæmatite is brought from Nassau and elsewhere. Both white and grey pig-iron of good quality are produced, as well as spiegeleisen; the first being employed for the production of steel in the puddling-furnace and open hearth, and the two last in the Bessemer process. The older furnaces were generally very small, and were worked exclusively with charcoal, but since the establishment of railway communication with the coal-fields of the Ruhr basin, these have been generally superseded by furnaces of larger capacity, in which coke and the hot-blast are employed. This has resulted in greater regularity both in the quantity and quality of the metal produced, in addition to which the high temperature in the region of the hearth, resulting from the hot-blast, is found to increase the quantity of manganese reduced.

The state of oxidation in which manganese exists in the ore also exercises considerable influence on the amount of that metal contained in the pig-iron made; when spathose ores are employed, the resulting pig-metal is more highly manganiferous than when hæmatites containing manganese are made use of.

At Charlottenhütte the charges for spiegeleisen have the following composition:—

| | Cwts. | |
|---|---|---|
| Roasted spathic ores | 28·8 | yielding 44 to 45 per cent. of pig-iron. |
| Raw brown hæmatite | 7·2 | |
| ,, limestone | 9·0 | |
| Total | 45·0 | |
| Coke | 20·0 | |

The coke contains 8 per cent. of ash; number of charges daily, 36; produce, 30 tons; consumption of coke per ton of metal, 22 to 23 cwts.; number of tuyers, 3; back 3 inches, and sides 3⅜ inches, diameter. Temperature of blast, 280° to 300° C.; pressure, 3⅜ lbs. per square inch.

In the Müsen district, spiegeleisen was formerly produced from the Stahlberg spathic ores without the addition of flux, but since it has been customary to add a certain amount of limestone the percentage of manganese in the pig-iron has been much larger than previously. At the Charlottenhütte the pig-iron produced usually contains 8 per cent. of manganese, but this is reduced to one-half when the blast is allowed to fall from 300° to 100° C.

The ores employed in South Staffordshire are partly clay ironstones from the Coal-measures ("native mine") and partly red and brown hæmatites from North Staffordshire, Lancashire, and elsewhere; the make is chiefly grey-pig for forge purposes. Forge-cinder is extensively employed in the production of common hot-blast metal; but the best mine pig is still made from coke with cold-blast. According to Mr. Jones, this district annually consumes 1,746,500 tons of ironstone, and 150,000 tons of forge- and mill-cinder. The coal of South Staffordshire belongs to the non-caking class, and is used partly in the raw state and partly coked; the coke, which is brittle, contains from 4·2 to 4·6 per cent. of ash, and from 0·3 to 0·5 per cent. of sulphur. The pressure of blast varies from 2½ to 3 lbs. per square inch, and the temperature from 300° to 330° C.; a small number of furnaces are worked with cold-blast.

The flux is principally Silurian and Carboniferous limestone; the average consumption of coal per ton of metal made is, in hot-blast furnaces, 55 to 60 cwts., and in cold-blast from 60 to 70 cwts., or rather its equivalent in coke. In addition to this, 2 cwts. of coal are necessary for the calcination of the ores, and from 15 to 22 cwts. are required in the hot-blast stoves, as the waste gases escaping from the throat of the furnace are but seldom utilised in the district. The produce of the furnace is from 120 to 150 tons weekly, although some of the largest yield from 180 to 250 tons of metal per week.

The furnaces of the Cleveland district, which are remarkable for their large dimensions, are worked entirely with the hard compact coke from South Durham, containing from 4 to 10 per cent. of ash and from ½ to 1 per cent. of sulphur. The ores treated are principally the argillaceous carbonates of the district, to which a little red hæmatite is sometimes added. It contains from 26 to 33 per cent. of metallic iron, which is increased to 37 or 40 per cent. by calcination.

The flux is limestone, derived chiefly from the Pennine range, containing, in its raw state, from 87 to 96 per cent. of calcium carbonate.

In order to produce a ton of grey foundry-pig from Cleveland ironstone without admixture of foreign ores or of mill-cinder, from 19 to 28 cwts. of coke and from 10 to 14 cwts. of limestone are required; the amount in each case varying in accordance with the quality of the ore and fuel, and the regularity of the working of the furnace, &c.

In one of the furnaces at Newport, near Middlesborough, described by Sir B. Samuelson in a paper read before the Institution of Civil Engineers, the average consumption of fuel, excluding the six weeks immediately after blowing-in, was 20·35 cwts. per ton of iron produced; the minimum quantity used in any one week 18·78 cwts., and the maximum quantity 22·12 cwts. per ton of iron. The average quantity of calcined ironstone used was 46·11 cwts. per ton of iron, the minimum quantity used in any one week 44·16 cwts., and the maximum quantity 48·04 cwts. per ton of iron. The average quantity of limestone employed was 10·71 cwts., the minimum quantity in any one week 10·35 cwts., and the maximum quantity 11·26 cwts. per ton of iron made. The average weekly production of pig-iron was 430 tons, and the maximum 466 tons, but the produce per furnace was subsequently increased to 500 tons. This furnace is 85 feet in height, 25 feet in diameter at the boshes, and has a capacity of 30,000 cubic feet; the consumption of fuel and flux, in proportion to the production, is about 15 per cent. less than that in four other furnaces erected in 1863–64, of which the internal capacity is only 16,000 cubic feet. Three furnaces erected at South Bank, in Cleveland, in 1854, have each a capacity of 5,079 cubic feet, and consume from 32 to 40 cwts. of coke, and from 14 to 15 cwts. of limestone to the ton of iron; the weekly production varies from 120 to 160 tons.

Among the largest furnaces in Cleveland are two at Ormesby. No. 1 is 90 feet high, 28 feet broad in the boshes, 23 feet at the top, 10 feet in the hearth, and of 33,400 cubic feet capacity. No. 2 is of the same height, but only 28 feet in the boshes; but as these are lower down, the capacity is somewhat larger, or 35,051 feet; the hearth is only 8 feet in diameter. It was found that these furnaces worked better, giving a larger production, with a smaller proportional consumption of fuel, when the overhang of the tuyers in the hearth was reduced so as to give a greater distance between their ends. Thus with 6 feet distance the consumption was 19·94 cwts. of coke per ton of No. 3 pig-iron, but when the distance was increased to 7 feet 4 inches the fuel was reduced to 19·75 cwts., and when the blast was heated to 1,465° F. to 18·67 cwts. No perceptible improvement was, however, obtained when the distance was increased to 8 feet. The weekly make under the latter condition was 566 tons. A smaller furnace of 20,000 feet capacity, which made

483 tons with 603 tons of coke, with the tuyers 6 feet apart, was brought up to 599 tons, with 630 tons of coke, by increasing their distance to 7 feet, the blast-temperature being increased from 1,321° to 1,430° F. at the same time.

The most productive furnaces, as regards their weekly make of pig-iron, are those treating the rich hæmatites of Lancashire and Cumberland. In the earlier days of the Barrow district, furnaces 56 feet in height and 16½ feet diameter at the boshes, with a cubic capacity of 9,500 feet, required, according to Jordan (1864), the following materials for the production of one ton of pig-iron:—

| | Cwts. |
|---|---|
| Red hæmatite, unroasted | 34 to 34½ |
| Coke from Durham | 18 to 18½ |
| Limestone | 5¾ |
| Slack for stoves | 3 |

The gases were generally collected, and exclusively employed in firing steam-boilers. About 7,000 cubic feet of blast, heated to 350° C., were supplied per minute through six 3-inch tuyers, at a pressure of 2½ lbs. per square inch. Under these conditions the maximum production appears to have been at the rate of about 630 tons weekly, but the average produce did not exceed 575 tons.

Most of the furnaces of this district have now been raised to 70 or 80 feet in height, fire-brick stoves have been erected in place of those with iron pipes, and the make has been brought up to 700 or 800 tons per week. The average production of all the West Coast furnaces in 1885 was 548 tons per week. At Consett 800 tons per week are made in furnaces 55 feet by 20 feet, with Whitwell stoves, smelting Bessemer pig-iron from hæmatite, with 19½ cwts. of coke per ton of metal.

The furnace, figs. 52 and 53, when newly started in 1872, made 300 tons per week, with a consumption of 26 cwts. of Lancashire coke per ton of metal; blast, 7,760 cubic feet per minute, at a temperature of 425° C., and pressure of 4 lbs. per square inch.

The charge containing, on an average, 50 per cent. of metal, was made up as follows:—

| | |
|---|---|
| Red hæmatite | 2 parts. |
| Irish ore | 1 part. |
| Spanish ore, Bilbao | 1 part. |
| Staffordshire red mine | 2 parts. |
| Purple ore | 1 part. |

The purple ore, obtained from various works in the neighbourhood at which copper is extracted by the wet process from burnt Spanish pyrites, contains, on an average, 67 per cent. of metallic iron.

At Bochum, in Westphalia, Bessemer iron is smelted from mixtures of Bilbao, Algerian, Swedish and Nassau magnetite and hæmatite, and Siegen spathic ore, with a very pure coke containing only 4·8 per cent. of ash. The furnaces, 72 feet by 20 feet, of about 15,000 cubic feet capacity, are supplied with 600 cubic feet of blast per minute, heated in Whitwell stoves to 700° or 780° C. The average production is 123 tons of pig-metal daily per furnace, with a consumption of 20·4 cwts. of fuel per ton.

*Coal in the Blast-Furnace.*—The principal ores employed in the Welsh blast-furnaces are "native mine," chiefly argillaceous carbonates with some blackband; brown hæmatites from Llantrissant, Forest of Dean, Northamptonshire, and Spain; red hæmatite from Cumberland, with occasionally a little from the Island of Elba, and spathic carbonates from Somersetshire.

In the eastern district the fuel employed is partly coal and partly coke, the latter being exclusively used in furnaces working on cold-blast. In the neighbourhood of Swansea a small number of furnaces are worked with anthracite. Forge- and mill-cinders are largely used in the production of white forge-pig.

According to Truran, some thirty years ago, the foundry-iron furnace at the Dowlais Works had a capacity of about 7,500 cubic feet, and was blown with a blast of 5,390 cubic feet of air per minute. For every 20 cwts. of crude iron obtained, 48 cwts. of calcined ore, 50 cwts. of coal, and 17 cwts. of limestone were required. The weekly production of iron was about 130 tons.

The consumption of materials per week was as follows:—Calcined ore, 312 tons; coal, 325 tons; limestone, 110½ tons; air supplied weekly by the tuyers, 1,695 tons.

For the production of white-iron for the forge, in furnaces of similar capacity to the foregoing, a larger amount of blast with a different burden was employed.

The consumption of solid materials to a ton of crude iron averaged 28 cwts., calcined argillaceous ore, 10 cwts., hæmatite, 10 cwts., forge- or refinery-cinder, 42 cwts. of coal, and 14 cwts. of limestone, with a volume of 7,370 cubic feet of blast per minute; the weekly production of crude iron was 170 tons. In this case the consumption of solid materials per week was 884 tons, and the weight of air injected by the blast 2,318 tons.

For the production of iron of inferior quality for the forge, the burden was composed of the following materials:—Hæmatite, 16 cwts., refinery-cinder, 25 cwts., coal, 36 cwts., and limestone, 16 cwts. per ton of crude iron. The capacity of the furnace and the volume of the blast were the same as in the last instance. From this furnace a weekly production of 190 tons of crude iron was sometimes obtained, the consumption of solid materials being 883 tons. The estimated time of the descent of a charge was from forty to forty-six hours.

At the same works in 1863, the consumption of coal per ton of mine pig, from variable mixtures of argillaceous ore and brown and red hæmatite, had been reduced to from 23 to 27 cwts.; the make of the furnace ranging from 172 to 280 tons per week. The larger production, in relation to the amount of fuel consumed, chiefly resulted from the greater richness of the charge arising from the larger amount of red ore used.

The newer furnaces are considerably larger than those formerly employed, and, when working on white-iron, their production ranges from 250 to 300 tons weekly. The use of the cup-and-cone charger is now general, the waste gases are economised, and hot-blast is employed.

The Scotch furnaces each produce, on an average, about 200 tons of pig-iron weekly; but in the blackband districts it is said to be sometimes as high as 270 tons.

At Irthlingborough Iron-Works, Northamptonshire, furnaces, 55 feet by 18, with 7 feet hearth, and 9,025 cubic feet capacity, make 342 tons weekly, smelting Northamptonshire ore in the proportion of four parts of raw to one part calcined, with an average produce of 40 per cent. The fuel used is about one-half hard Derbyshire and Yorkshire coal, three-tenths soft coke, and one-fifth hard Durham coke, the consumption being at the rate of 23·13 cwts. per ton, besides 1·5 cwt. of slack, used for boilers and calcining. The average burden is 50 cwts. raw ore, 8·33 cwts. calcined ore, and 14·13 cwts. oolitic limestone per ton.

*Anthracite Furnaces*, &c.—Anthracite is employed for the production of pig-iron in South Wales and, to a much greater extent, in the United States of America.

Anthracite is liable to decrepitate when strongly heated, and some varieties, when suddenly exposed to a high temperature, become reduced to a state of absolute dust. In furnaces in which anthracite is the fuel made use of, the accumulation of these small particles sometimes becomes so great as to materially obstruct the passage of the blast. When this takes place it is usual to cease charging and to continue the blast, when, as the solid materials descend, the fine particles of anthracite are blown away. A greater difficulty, however, experienced in the use of anthracite, arises from the running together of the slag and decrepitated particles of fuel into infusible masses, which are liable to cause the furnace to become obstructed.

In the year 1863 the anthracite furnaces at Yniscedwin, South Wales, were from 25 to 30 feet only in height, since a low stack is less liable than a high one to become obstructed by the small particles of fuel resulting from decrepitation.

Other anthracite furnaces in the same district were from 36 to 40 feet in height, and were blown at a pressure of from 4 to 6 lbs. per square inch, with air heated to from 320° to 450° C. The consumption of air amounted to 6,000 cubic feet per minute, the weekly production was 80 tons, and the consumption of anthracite $2\frac{1}{2}$ tons for each ton of pig-iron.

The anthracite furnaces of Pennsylvania are worked with a large number of tuyers and with a pressure of blast varying from $6\frac{1}{2}$ to $7\frac{1}{2}$ lbs. per square inch. The ores treated are usually massive magnetites and hæmatites, containing from 50 to 60 per cent. of iron.

According to Mr. J. M. Hartmann, the newest type of anthracite furnace in the United States is 75 feet high, 70 feet to the top of the column of materials or *stock-line*, 31 feet to the top of the boshes, and 6 feet to the tuyers; the corresponding diameters being 17 feet at the stock-line, 18 feet in the boshes, and 10 feet 8 inches in the hearth. The charging is effected by two concentric bells 6 and 11 feet in diameter, which distribute the charge upon two rings respectively 4 and 11 feet in diameter. The water-cooled *cinder-notch* is 3½ feet above the hearth bottom, and 2½ feet below the tuyers. The latter overhang 10 inches, leaving a clear space between their ends on opposite sides of 9 feet. The hearth walls are 24 inches thick, and are completely surrounded by a water-jacket, which extends from 8 inches below the hearth bottom to 9 feet above the tuyers. The effective volume of blast is 14,400 cubic feet per minute, at a pressure of 4 to 8 lbs., and is heated to 850° F., if pipe-stoves are used, or to 1,200° F. in brick stoves. Seven tuyers of 4½ inches diameter are used. The coal is supplied in lumps 4 inches cube. Owing to the tendency of anthracite to decrepitate and crumble when strongly heated, it is necessary to have great reserve of blowing power, and at times a pressure of 20 lbs. may be required to overcome the resistance of the mixed dust of ore, limestone, and fuel, which forms at the sides of the furnace, and must be removed through the tuyer-holes. A very broad hearth is also essential in order to allow space for the oxidizing action of the air upon the fuel. Anthracite shows but little tendency to reduce carbonic acid in its passage through the furnace, and therefore the fuel charged arrives at the tuyers almost unchanged. When smelting siliceous limonites of 43 per cent., and magnetites of 54 per cent. produce, in the proportion of $\frac{5}{8}$ of the former to $\frac{3}{8}$ of the latter with a dolomitic flux, the average results obtained were :—

| | |
|---|---|
| Coal per ton of iron (Nos. 1 and 2) . . | 28 cwts. |
| Ore „ „ „ . . . | 47½ „ |
| Flux „ „ „ . . . | 25½ „ |

With blast at 850° F. the maximum weekly make was 600 tons, but with fire-brick stoves at 1,200° F. the coal is reduced to 24 cwts., and the out-put increased about 20 per cent. The temperature of the escaping gases averages about 250° F. Rapid driving is of great importance in anthracite smelting, in order that the fuel may be brought to the tuyers with as much as possible of its occluded gases, since otherwise it is very difficult of combustion. The breadth of the hearth is also a factor of great importance. Thus a furnace 20 feet in the boshes with a 7-feet hearth made only 350 tons weekly, but was brought up nearly to 600 tons weekly by enlarging the hearth to 10 feet.

The same principle of obtaining a large out-put from a furnace by the use of a very large volume of blast is now adopted at many works in the United States, smelting hæmatite ores with coke for Bessemer pig-iron. At the South Chicago furnaces, which are 20 feet wide in the boshes, 11 feet in the hearth, and 15 feet 4 inches at the top, with a total height of 75 feet, and 15,000 cubic feet capacity, 209 tons of metal are cast daily from a mixture of ores averaging 63·75 per cent. of iron. Each furnace has three 60 × 21 feet Whitwell stoves, heating the blast to 1,450° F. The blowing power is very large, namely, two 84-inch cylinders, 4½ feet stroke, running at a speed of 252 feet per minute, and delivering 19,320 cubic feet of air per minute at 6 lbs. pressure, reduced to 4 lbs. at the tuyers. The latter are seven in number, and 7 inches in diameter. The average charges are, per ton of pig made—

| | |
|---|---|
| Ore . . . . . . . . . | 31¾ cwts. |
| Limestone . . . . . . . | 5¼ „ |
| Coke (11·61 per cent. ash) . . . . | 16⅔ „ |

The weight of blast per ton averages 62⅔ cwts., and that of the waste gases 86¾ cwts.; the latter contain carbonic oxide in the ratio of 46 volumes to 100 volumes of carbonic acid. Their average temperature was 480° F. when the blast was at 1,375° F. The work done by the blowing-engine is 508 HP., and about half as much more power is required to work hoisting- and pumping-engines.

The largest out-put from any single furnace was obtained at the Lucy No. 2 furnace at Pittsburg, which has produced 345 tons in one day, 1,975 tons in a week, and 8,035 tons in a month. This is, however, considered to be too high a speed, and less advantageous than that given above.

Spiegeleisen is made at Newark, New Jersey, from residues obtained in the treatment of a mixture of red zinc ore, franklinite, and willemite, from the production of zinc oxide to be employed as a pigment. The residue from the furnaces in which the volatilization of zinc is effected is a black cindery mass, containing the whole of the iron and manganese of the franklinite, and the silica of the willemite, together with some oxide of zinc. It is estimated to contain about 25 per cent. of iron, and is smelted with anthracite in furnaces of small dimensions. Their height is usually only 20 feet, and their diameter at the boshes 7 feet; the blast is introduced at a temperature of 200° C., and with a pressure of 4 lbs. per square inch; limestone is employed as flux, and the weekly make is about 25 tons. The waste gases are consumed in stoves for heating the blast, but before they can be thus employed the oxide of zinc, resulting from the oxidation of that metal still retained by the residues, and volatilized in the furnace, requires to be separated by a system of wrought-iron condensers. This oxide is from time to time removed, and not being sufficiently pure to admit of being employed as paint, is sent to spelter furnaces for reduction. The consumption of fuel is at the rate of 3¼ tons per ton of pig-iron produced.

*Ferro-Manganese.*—In the production of ferro-manganese the highest attainable temperature is required in the hearth of the furnace, the charges should be kept as free as possible from silica, and the slags must be basic without containing a large excess of lime. In this case, therefore, magnesia is useful, and for the most manganiferous metal barytes and fluor spar are sometimes added. The proportion of manganese in the slag may be about 8 per cent. of MnO for 20 per cent. spiegel, 15 per cent. for 70 per cent., and from 17 to 20 per cent. for 80 per cent. ferro-manganese. It is important not to push the reduction of manganese too far, as the slags then become too calcareous, and silicon is reduced, preventing the satisfactory formation of spiegel.

There is also a notable proportion of manganese lost by volatilization, which, together with the zinc oxide and alkalies in the ore, are carried over by the gases, and may be very destructive to the fire-brick stoves if not previously passed through a washer or dust-catcher.

At Terre-Noire, where the first high ferro-manganese was made, following Henderson's practice in making the same alloy in the Siemens furnace, the hearth was built of bricks moulded in gas carbon. These are found to be more durable than fire-bricks, and may be easily kept cool by watering the outside, since the thermal conductivity of carbon is about twelve times that of fire-clay. By using blast at 750° C. and rich ores, metal with more than 80 per cent. of manganese may be obtained. A practical rule for fluxing is to give twice as much lime and one-half as much baryta as there is silica present in the ore and fuel. At Terre-Noire the charges for 82 per cent. metal were as follows:—

| | | | | |
|---|---|---|---|---|
| Huelva ores, | pyrolusite | 9·6 cwts., | containing 0·28 cwts. Fe., | 5·04 cwts. Mn. |
| Almeria „ | „ | 4·0 „ | „ 0·06 „ | 2·00 „ |
| Tafna „ | hæmatite | 0·4 „ | „ 0·22 „ | ... „ |
| | | 14 cwts. | 0·56 | 7·04 |
| | | | 25 per cent. loss in slag . | 1·76 |
| Limestone . . | | 4·4 | Manganese . . . | 5·28 |
| Sulphate of barium | | 1·2 | Iron . . . . | ·56 |
| | | | Carbon and silicon . | ·46 |
| | | 19·6 | | 6·30 |

Yield of ore, 45 per cent.; of charge, 32 per cent.

The furnace was blown with two $3\frac{1}{2}$-inch tuyers, with blast at $2\frac{1}{2}$ lbs. pressure, heated to 680° to 750° C., being driven at only half the speed required for Bessemer pig-iron. The daily make was $10\frac{3}{4}$ tons, averaging from 81 to 85 per cent. manganese, with a consumption of 54 cwts. of coke per ton. The proportion of manganese reduced varied from 70 to 79 per cent. The gas of a furnace working on high ferro-manganese is of low heating power, as the oxygen given off by the ore when first heated burns carbonic oxide to waste in the throat of the furnace. Thus in making 60 per cent. ferro-manganese, the gas only contained 5·5 of carbonic oxide to 30 per cent. of carbonic acid, and was practically incombustible. Subsequently about 10 per cent. of bituminous coal was added to the coke to furnish gas for the boilers. This inconvenience may be, to some extent, remedied by previously exposing the manganese ore to a low heat in a calcining-furnace, which reduces $MnO_2$ to $Mn_3O_4$.

When the zinciferous ore of Cartagena is used, hard deposits form in the gas-flues, which must be cleaned out at intervals of about seven days. These consist essentially of oxide of zinc, the average metallic contents being manganese 3·34, zinc 63·40, iron 1·56 per cent. The dust, on the other hand, is much richer in manganese, containing manganese 23, iron 6, and zinc 1·36 per cent. It is also remarkable as containing 11 per cent. of baryta, mainly as silicate, which shows that the sulphate of barium in the flux is entirely decomposed before reaching the hearth. No portion of the sulphur is taken up by the metal, it being mainly eliminated as sulphides of calcium and manganese in the slags, and to a smaller extent as sulphurous acid in the gases.

High ferro-manganese oxidizes very readily, and it is therefore necessary to stop the blowing-engine when casting, otherwise there may be a notable difference in the percentage between the first and last beds of the cast. Very hot blast is not necessary in making ferro-manganese if the use of siliceous ores be avoided and the slags be rendered sufficiently basic with lime.

### *Heat Absorbed for Work Done in Blast-Furnaces.*

The chemical phenomena of the blast-furnace have been very completely examined, both analytically and experimentally, by many observers, among whom Sir I. L. Bell, Ritter P. v. Tunner, the late Professor Gruner, and Professor R. Åkerman may be more particularly mentioned for the importance of their contributions to this subject. The works of Sir I. L. Bell, the titles of which are given below,[1] should be consulted by the reader desiring fuller information on this subject, which can only be here noticed in a general way. The work of the furnace consists chiefly in the reduction of the ore in its descent by successive steps from ferric oxide, through magnetic oxide, and ferrous oxide into metallic iron, by carbonic oxide produced from the direct combustion of fuel in the hearth with heated air. The accessory reduction of the oxides of manganese, phosphorus, sulphur, and silicon, which, under certain conditions, combine with the reduced iron, can only be effected by contact with solid carbon, and therefore takes place chiefly in the lower region of the furnace below the boshes. Other operations, requiring heat only, are the expulsion, in the higher parts of the furnace, of water from hydrates and moist ores, and of carbonic acid from the flux; also, where raw coal is used, its conversion into coke, and the fusion of metal and slag in the hearth. A notable portion of the heat is carried off by the escaping gases, and lesser amounts by the tuyer-water and by radiation from the furnace walls. According to Sir I. L. Bell, the relative calorific importance of these factors may be estimated as follows in smelting Cleveland iron ore for No. III. pig, under the ordinary conditions of working, and producing 28 cwts. of slag per ton of metal. The figures are expressed in 20 weight units (either kilogrammes or cwts.), and the corresponding heat units in Centigrade degrees.

---

[1] 'Chemical Phenomena of Iron-Smelting:' London, 1872. 'Principles of the Manufacture of Iron and Steel:' London, 1884.

| | Weight in Cwts. | Heat Units per Cwt. | Total Heat Units. |
|---|---|---|---|
| Reduction of $Fe_2O_3$ to Fe | 18·60 | 1,780 | 33,108 |
| Carburizing reduced iron | ·60 | 2,400 | 1,440 |
| Evaporation of water in coke | 0·58 | 513 | 313 |
| Reduction of $P_2O_5$, $SO_3$, and $SiO_2$ | ... | ... | 3,500 |
| Fusion of pig-iron | 20·00 | 330 | 6,600 |
| Calcination of flux | 11·00 | 370 | 4,070 |
| Reduction of $CO_2$ of flux to CO | 1·32 C | 3,200 | 4,224 |
| Decomposition of water in blast | 0·05 H | 34,000 | 1,700 |
| Fusion of slags | 27·92 | 550 | 15,366 |
| | | | 70,321 |
| Transmission through walls of furnace | | | 3,600 |
| Carried off in tuyer-water | | | 1,800 |
| Expansion of blast, transmitted to ground, &c. | | | 3,389 |
| | | | 79,110 |
| Carried off by gases | | | 7,900 |
| Total heat requirements per 20 cwts. | | | 87,010 |

The heat required for these different operations is derived from two sources, the larger part being developed by the combustion of fuel in the hearth, and the remainder introduced by the hot-blast. As regards the former, it is not possible to oxidize the carbon of the fuel beyond the ratio of two CO to one $CO_2$[1] in the waste gas, under which condition the heat developed is $\frac{1}{3}$ 8,000 + $\frac{2}{3}$ 2,400 or 4,266 units per unit of carbon; or, assuming the fuel to contain 10 per cent. of volatile and incombustible matters, $0·9 \times 4,266 = 3,840$ units per unit of coke. If the blast is heated to 1,004° F. or 540° C., the heat introduced may be estimated at 12,000 units, leaving 75,000 units to be provided by the combustion of coke, which, on the estimate above given, corresponds to 19·53 cwts. This supposes the weight of blast to be about five times, and that of the waste gases nearly seven times that of the iron made. Under these conditions, which are somewhat more favourable than those realized in the most advantageous working on the large scale, about 86 per cent. of the heat is furnished by the fuel, and 14 per cent. by the blast; the latter being derived from the combustion of the waste gases, which furnish a further amount of heat, estimated at 28,000 units, for raising steam for the blast, lift, and pumping-engines required in working the furnace, while about 11,000 units are available for other purposes. As the total heat obtainable by the oxidation of the coke to $CO_2$ is about 150,640 units, while that usefully accounted for is about 110,000 units, the thermal efficiency realized is about 74 per cent., or considerably higher than that of any other industrial process for which high temperatures are required.

---

[1] This is explained by the fact that carbonic oxide has no reducing efficiency when diluted beyond this extent with carbonic acid, at the temperature prevailing at the top of the furnace.

In the table given at p. 271 the items concerned in the reduction and carburization of the metal may be taken as constant for all kinds of anhydrous or calcined ores, while the remainder will vary with the richness of the latter and the amount of siliceous and earthy waste to be fluxed. Thus in the case supposed, the fusion of these slags requires about one-sixth of the total heat (87,000 units), while in the South Chicago furnaces, under the conditions noted on pp. 267–8, making only 9·27 cwts. of slag per ton, the proportion is reduced to one-fourteenth of the smaller total of 70,000 units. This difference is still more apparent when charcoal is used, as then the weight of slag is reduced to that strictly required for fluxing purposes, the fuel being free from sulphur; whereas in smelting even the purest ores with coke, a notable quantity of slag is necessary to prevent the iron becoming sulphurous, which is likely to happen if the sulphur in the slag exceeds 2 per cent. This is, perhaps, one of the principal causes why a smaller weight of fuel is required, as compared with coke, when charcoal is used for smelting similar ores; but the complete answer to this question has still to be worked out.

The possibility of economizing fuel by augmentation of the temperature of the blast can, upon theoretical grounds, be shown to be strictly limited, and that after a certain point no further saving is likely to be realized by higher heating in the stove. Up to the present time, however, this point has not been reached in practice, although the margin for saving is constantly diminishing, the latest forms of Cowper's stove, giving a heat of 1,400° F. (760° C.), having effected a notable saving as compared with those heating the blast to 1,200° or 1,300° F.

The quantity of fuel required, per unit of make, varies not only with the yield of the ore and amount of slag, but also with the quality of the metal produced, about 10 per cent. more coke being required for making No. I. foundry iron than for No. III. The following estimate, made by De Vathaire, gives the fuel necessary per ton of different classes of metal smelted from ores of about 40 per cent. produce:—

| | |
|---|---|
| No. I. foundry pig . . . . | 24 cwts. |
| No. II. " . . . . | 23 " |
| No. III. " . . . . | 22 " |
| No. IV. " . . . . | 21 " |
| No. V. " . . . . | 20·5 " |
| Mottled " . . . . | 20 " |
| White, crystalline (Weissstrahl). | 19·5 " |
| White, common . . . . | 19 " |

The above are for ordinary pig-irons; when manganiferous, 0·3 cwt. per unit of manganese reduced must be added to the corresponding number in the above table. Thus a 20 per cent. spiegel requires 20 × 0·3 = 6 cwts. more than Weissstrahl, or 25·5 cwts., and 50 per cent. ferromanganese 50 × 0·3 = 15 cwts. more than No. III., or 37 cwts. in all.

## CONVERSION OF GREY CAST-IRON INTO WHITE.—REFINING.

This process, which is now comparatively unimportant, being used only in a few localities in making special qualities of iron, was in the earlier days of puddling essential to the correct working of the operation. Its object is the removal of silicon from grey pig-metal, in order that the labour of the puddler and the duration of the fining operation may be reduced; but since the introduction of iron bottoms to the puddling furnace and the use of ferric oxide as fettling, grey metal is now generally puddled direct for common bar-iron, the extra consumption of materials being more than compensated by the saving of fuel, which is considerable, as only the best coke or charcoal could be used. Refining is now principally confined to forges making the best brands of malleable iron in Yorkshire, and to tin-plate forges in South Wales. A modification of some importance was introduced some years since by Sir I. L. Bell and Baron Krupp for the improvement of phosphoric pig-iron for steel-making; but this has been abandoned since the development of the basic Bessemer process.

When grey cast-iron is fused in an oxidizing atmosphere, the silicon present is oxidized, and, becoming silica, unites with a portion of the iron, oxidized at the same time, to form a fusible ferrous silicate. If the metal be now run into moulds and suddenly cooled, a peculiarly white iron is produced, which is analogous in composition to that smelted from pure ores at a low temperature, with a heavy burden of material. The same result may be obtained by throwing water on the surface of a bath of molten metal and subjecting the thin plates thus obtained to the action of air at a red heat during several hours; this method is employed in various parts of Germany. The more usual process, however, consists in melting the metal with coke or charcoal in a rectangular hearth, provided with tuyers more or less inclined, through which air is blown upon the surface of the fused metal.

In this country the refinery usually consists of a strong cast-iron framework, supporting a low brickwork chimney, A, figs. 78 and 79, which represent, respectively, a transverse section and plan of a refinery erected at the Dowlais Iron-Works. The bottom rests upon a floor of dressed sandstone, *a*, about 12 inches in thickness, which is supported upon a foundation of brickwork or masonry. At each side, and at the back, within the vertical frames, are fixed cast-iron water-blocks, *b*, while a dam-plate, *c*, fig. 79, of the same material, closes the front; thus forming a quadrangular cavity about 4 feet square, inside measure, and 15 to 18 inches in depth. Above the side-blocks, and resting on a ledge cast for their reception, are the tuyer-plates, *d*, about 2½ inches in thickness, provided with openings for the insertions of water-tuyers, and bolted at their ends to the vertical framing. In front, resting on the dam-plate, it is usual to have another plate, for the convenience of filling and working the fire. A little above this, in front, and also above the water-block in the rear, cast-iron doors 2½ feet high are hung to the side-frames.

At a sufficient distance from the floor of the refinery, and a little in advance of the dam-plate, is placed the casting-bed or pig-mould, B. A brickwork or cast-iron cistern, about 30 feet long, 4 feet wide, and 2 feet deep, is laid at right angles to the axes of the tuyers, and is slightly inclined from the hearth. On this rests the casting-bed, composed of thick cast-iron blocks, *e*, 3½ feet square and 6 to 8 inches in thickness, having flanges on each side by which they are supported on the edges of the cistern beneath, and a depression in the upper surface to confine the liquid metal within the desired limits. This cistern is filled to within an inch of the under surface of the mould-blocks with water, which is maintained at this level by a small stream which constantly flows in and escapes by an overflow-notch. The mould-blocks are provided with rebated joints luted with fire-clay, and are maintained in close contact with each other by clamps fitting corresponding snugs cast on the sides of the moulds. They are often made with a rib running down the centre, dividing the mould longitudinally in two parts, so as to reduce the labour of breaking up the plate of refined metal.

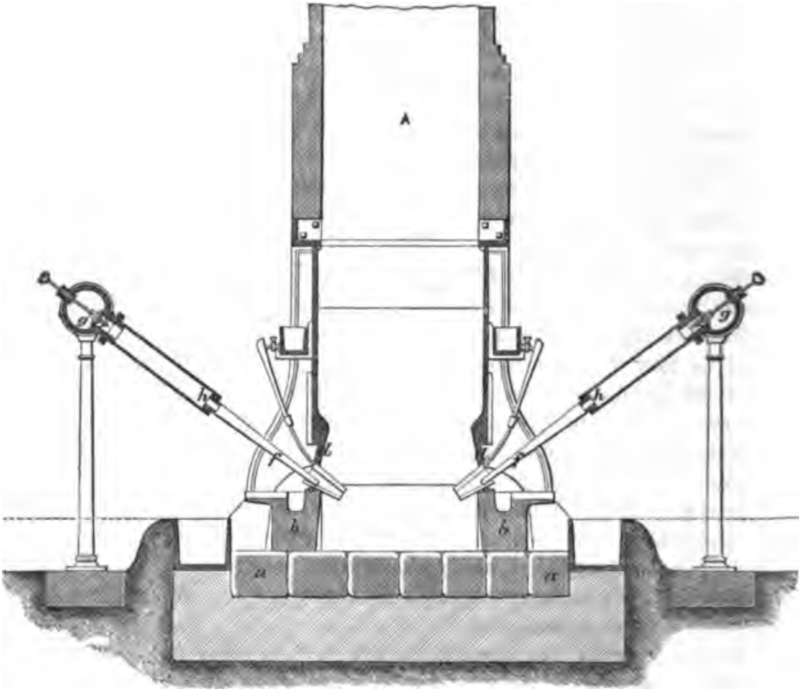

Fig. 78.—Refinery; transverse section.

The blast is usually supplied through two or three small nozzles on either side of the hearth, each pipe being provided with a stop-valve, *g*, for regulating the quantity. The connection between the movable nozzles, *f*, and fixed supply-pipes may be made either by means of short leathern hose fastened at their ends by screw-clamps, or by ball-and-socket joints, *h*; the former method is the cheaper, but the latter the more durable.

Refineries are also sometimes constructed with a single nozzle at the back, in which case the framework, water-blocks, moulds, and all the other parts are made much lighter, and the fire-place is of smaller dimensions. They are often distinguished as melting-down and running-in refineries; cold pigs from the blast-furnace, old castings, scrap, &c., are melted in the former, while liquid metal is allowed to flow into the latter directly from the furnace.

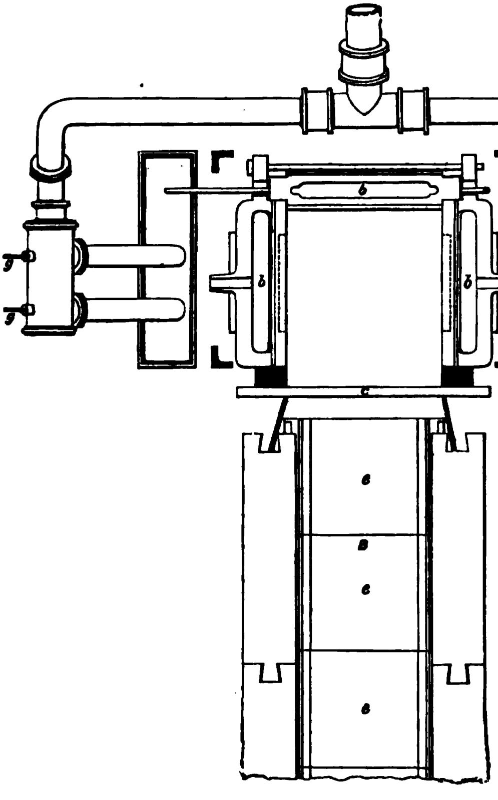

Fig. 79.—Refinery; plan.

The melting-down refinery is usually placed in a building at some distance from the blast-furnace, while the running-in refinery is, on the contrary, generally built immediately contiguous to it; this method of working, which effects a considerable saving of coal, was first introduced at Dowlais.

The operation of refining crude pig-iron is usually conducted in the following way. The floor of the hearth is strewed with broken sandstone and a fire is lit in the centre; a quantity of coke is now added and a light blast directed upon it. A charge of pig-iron, scrap, or broken castings is now piled on the coke, a fresh quantity of fuel is heaped upon the metal, and the full power of the blast turned on. The weight of crude iron constituting a charge varies in accordance with the dimensions of the refinery, but it may, on an average, be taken at 2 tons, requiring the consumption of 5 cwts. of coke for its conversion into refined metal.

The broken sandstone on the floor fuses and glazes the surface of the hearth, while, in the course of an hour, the metal begins to melt, and dropping through the coke, reaches the bottom. In the course of from two hours to two hours and a half, the whole of the iron has become melted and lies under the coke, but the blast is still kept up and fresh coke is added. By this means a portion of the metal becomes oxidized, and uniting with the siliceous matter contained in the fuel, as also with the silica resulting from the oxidation of the silicon present in the cast-iron, forms a fusible vitreous slag. This slag, which is extremely rich in oxide of iron, exercises a strong decarburizing action on the iron on which it floats; but in order that these changes may be properly effected, the air from the different tuyers is allowed to play on the surface of the fused mass for a considerable time after the whole of the iron has collected at the bottom of the hearth. During this period the fuel is observed to be continually lifted by the motion caused in the metal by the escape of carbonic oxide gas produced by the reaction on the cast-iron of the rich silicates of iron constituting the slag. When the decarburization is judged to be sufficiently advanced, the tap-hole is opened, and the contents of the hearth are allowed to flow into the moulds, where they are cooled by a large quantity of cold water, by which treatment the metal is rendered extremely brittle. The slags are now separated, and the fine metal broken into pieces convenient for transport to the puddling-furnace, where it is to be freed from the remainder of its carbon and converted into soft iron.

The working of the refinery is continuous, so that as soon as one charge of metal is run out, the hearth is again prepared for the reception of a fresh supply. The time occupied in refining each charge will average about three hours; but white-iron does not require to be blown so long as grey-pig, which frequently requires from three and a half to four hours before it is sufficiently refined.

With the running-in refinery the case is somewhat different, since the metal is run into the hearth in a fused state, and the time and fuel consumed in melting the metal in the ordinary refinery are consequently saved.

When taken directly from the blast-furnace, 22·3 cwts. of common forge-, or 21·1 cwts. of good grey-pig, are required to produce 1 ton of fine metal, and the expenditure of coke is about 2 cwts. In the melting-down refinery the loss of metal is somewhat greater, and the expenditure of fuel 20 per cent. larger.

The loss of weight in refining hot-blast iron is usually greater than that experienced in the treatment of cold-blast pig, and the metal produced from blackband is especially difficult of treatment, chiefly owing to its ready fusibility, which renders long-continued blowing necessary; this results in an increased waste, 24 cwts. of crude iron being required to produce 1 ton of fine metal.

The tuyers are usually from 1½ to 1¾ inch in diameter, and are inclined at an angle of about 38°. In the running-in hearth 94,000 cubic feet of blast are required per ton of metal treated, but in the melting-down refinery 136,000 feet are necessary for white-pig, and 153,000 cubic feet for grey-iron. The weekly production of a refinery working on white-iron is from 150 to 160 tons, and with grey from 80 to 100 tons. The operation of refining may be accelerated by the addition of basic silicates, such as the slags from re-heating furnaces or forge-scale. The action of the blast is in this case supplemented by the use of the flux; the carbon of the cast-iron is in part oxidized by the oxygen of a portion of the oxide of iron, an equivalent amount of iron being at the same time reduced to the metallic state. Lime may sometimes be advantageously employed for the removal of sulphur, but it has a tendency to render the slags comparatively infusible, and can therefore be used in moderate quantities only; the same result is more efficaciously produced by the presence of manganese.

The usual thickness of the plates of fine metal is about 3 inches; when freshly fractured the surface has a silvery-white colour, the lower part being compact, with a radiated or columnar structure, while the top is dull and cellular.

A modification of the ordinary refinery process is used in Carinthia, chiefly for iron intended for the manufacture of steel. Grey or mottled pig-iron is melted in a hearth lined with brasque, charcoal being employed as fuel. The slags are partially removed from the surface of the fluid iron and oxidizing fluxes added; water is then thrown upon the metal and the chilled crusts taken off. The consumption of charcoal is about 5 cubic feet per 100 lbs. of refined iron produced.

*Eck's Refinery.*—As used at Gleiwitz and Königshutte in Silesia, this apparatus consisted of a reverberatory furnace with a gas-producer and a heated top blast. The metal when melted down with a small addition of limestone was blown by two side tuyers, and the operation lasted from two to five hours, according to the product required. As the process was mostly intended for the production of strong foundry metals, the silicon was not entirely removed.

*Parry's Process.*—In this method, which was experimentally used in South Wales, but never adopted on a large scale, fused metal from the blast-furnace was subjected to the action of numerous jets of superheated steam in the hearth of a reverberatory furnace heated by a coal fire in the usual way. From 3½ to 4 cwts. of forge-cinder, containing 66 per cent. of iron, was used per ton of metal blown, and the iron reduced compensated for that oxidized in the removal of silicon. About one-third of the phosphorus was also oxidized. Seven tuyers or steam-jets of ⅜-inch diameter, supplied with steam at 30 lbs. to 40 lbs. pressure, refined 1 ton of grey-iron in half an hour.

The following analyses of metal and slag obtained by this process at the Ebbw Vale Works are by Noad :—

| | Pig-iron used. | Refined Metal. |
|---|---|---|
| C, graphitic . . | 2·40 | 0·30 |
| Si . . . . . | 2·68 | 0·32 |
| S . . . . . | 0·22 | 0·18 |
| P . . . . . | 0·13 | 0·09 |
| Mn . . . . | 0·36 | 0·24 |
| Slag . . . | 0·68 | ... |
| | Forge-cinder added. | Cinder run out. |
| S . . . . . | 1·34 | 0·16 |
| $P_2O_5$ . . | 2·06 | 0·13 |

*Heaton's Process.*—A process was introduced some years since by Mr. Heaton, of Langley Mill Iron-Works, which, although intended for the production of steel, is essentially a modification of refining. Instead of a blast of atmospheric air, the oxidizing agent employed is nitrate of sodium. The converter is a circular cupola enclosed in an iron casing, having a movable bottom, kept in its place by iron clamps; this bottom is filled with nitrate of sodium in the proportion of one-tenth the weight of the metal treated, and, in some cases, a small quantity of siliceous sand is added. In order to prevent it from floating to the surface of the molten metal without undergoing decomposition, this flux must be covered by a perforated plate of cast-iron, and, when the bottom is thus adjusted, fused cast-iron is introduced through a charging-hole at top. During the first five minutes, the action of the nitrate on the oxidizable matter present is slight, but on the melting of the covering-plate, which usually occurs after that interval, a violent ebullition takes place, and a bright yellow sodium flame escapes from the top of the chimney. After continuing for about a minute and a half, this action rapidly subsides; the bottom of the converter is then detached, and is, with its contents, removed on a truck placed beneath it for that purpose. The product of this operation, called "crude steel," is not sufficiently liquid to be run into ingots, and the contents of the converter are therefore turned out upon the floor, and broken into fragments of convenient size for further treatment.

This consists of sundry re-heatings, squeezings, and hammerings; or the broken fragments may be melted in crucibles in the ordinary way for the production of cast-steel.

The following analyses of the material operated on, and of the products obtained by this process, were published by the late Dr. W. A. Miller in 1868:—

| | 1. | 2. | 3. |
|---|---|---|---|
| C | 2·830 | 1·800 | 0·993 |
| Si, with a little Ti | 2·950 | 0·266 | 0·149 |
| S | 0·113 | 0·018 | trace |
| P | 1·455 | 0·298 | 0·292 |
| As | 0·041 | 0·039 | 0·024 |
| Mn | 0·318 | 0·090 | 0·088 |
| Ca | ... | 0·319 | 0·310 |
| Na | ... | 0·144 | trace |
| Fe, by difference | 92·293 | 97·026 | 98·144 |
| | 100·000 | 100·000 | 100·000 |

1. Cupola-pig. 2. Crude steel. 3. Steel-iron.

It will be observed that the "steel-iron" contains as much carbon as ordinary strong steel, and that the amount of phosphorus retained is four times greater than that contained in best Yorkshire iron, and fourteen to fifteen times as much as in Swedish Bessemer steel. This process, which does not appear to have been carried out on a manufacturing scale, was subsequently examined by M. Gruner and by Mr. G. J. Snelus, of Dowlais. The analyses of the latter chemist, who has had a large experience in investigations of this nature, show that the most marked effect of the use of the alkaline nitrate is the almost entire removal of silicon; this is probably to a great extent due to the production of silicates of sodium.

*Henderson's Process.*—Henderson's method for the production of malleable iron and steel from inferior brands of pig is dependent on the action of titanic oxide on fluor-spar, by which silicon, sulphur, and phosphorus are said to be almost wholly eliminated. It has never been adopted on the large scale.

DEPHOSPHORIZING REFINERY PROCESSES.—The removal of phosphorus from pig-iron, as a preliminary to puddling or other processes, has been effected by Sir I. L. Bell by subjecting molten metal from the blast-furnace to the action of magnetic oxide of iron, either melted or in a very highly heated state. The operation was performed either in a revolving water-jacket puddling-furnace, or in a trough of wrought-iron, lined with bricks of purple ore, 12 feet long, 3 feet wide, and 3 feet deep, placed under a brick arch, and mounted upon a central axis, about which it oscillated like the beam of a steam-engine. The interior was raised to a red heat by a gas-fire, and the oxide of iron introduced, either melted or sufficiently hot to melt when the iron was run in. About 10 cwts. of metal were heated with 4 cwts. of liquid oxide, but the temperature was kept below that of an ordinary puddling-furnace. The operation lasted from ten to twenty minutes, during which time the vessel was kept oscillating, so as to traverse the charge repeatedly from one end to the other. The distance thus travelled in about ten minutes was from 720 to 960 feet. The charge was then tapped into moulds, where the iron and cinder were separated as in the ordinary refinery. The metal resulting from this so-called "purifying process" was, like that of the refinery, almost freed from silicon, and from 80 to 90 per cent. of the original amount of phosphorus was removed, while the carbon was very slightly acted upon. The slags contained about 6 per cent. of $P_2O_5$

A similar process, used for some time by Krupp, was described by the late Mr. A. L. Holley under the name of "pig-washing." A Pernot gas furnace, with a circular hearth rotating about an axis slightly inclined from the vertical, was used. The charge of 5 tons of melted metal was heated with 20 per cent. of its weight of pure iron ore, and when it contained more than 1 per cent. of silicon, some limestone was added. About 6 per cent. of silica in the charge was necessary to give the requisite fluidity to the slag. The ore was first charged and then strongly heated; the iron was run in, the revolutions of the furnace being regulated to 11 per minute. As soon as carbonic oxide flames began to appear (usually in from five to eight minutes) the metal was run off. The result was similar to that given above; but as a more manganiferous metal was used, the silicon was entirely removed, together with from 75 to 80 per cent. of the phosphorus. Manganese was also completely oxidized and passed into the slag. The cost of the operation was computed at 10s. per ton of metal treated, including melting in the cupola.

These processes, though at one time promising, have lost their significance since the introduction of the method of dephosphorizing by lime in the Bessemer converter.

## PRODUCTION OF WROUGHT-IRON FROM CAST-IRON IN OPEN FIRES.

The various processes employed for the production of wrought-iron from cast are either conducted in open hearths, in which the pig-metal is melted and decarburized before the blast of an inclined tuyer, or the transformation is effected by puddling, by which the same operation is performed on the hearth of an ordinary reverberatory furnace. In both cases the reactions involved are precisely similar; graphitic carbon first passes into the non-graphitic or combined state, and is subsequently converted into carbonic oxide, either directly by the oxygen of the blast, or indirectly by the action of oxide of iron in the slags. In many instances this oxidizing agent is supplied by the iron itself, which is always to a certain extent oxidized by the air of the blast during the process of fusion, while in others it is directly added in the form of hæmatite, forge-scale, finery-cinder, &c.

The methods employed for the production of malleable iron in open-hearths, although, from their antiquity and comparative simplicity, of much interest, are nevertheless gradually diminishing in importance; this arises from the rapidly extending use of the puddling-furnace, which can not only be employed with fuel and materials of lower quality, but also admits of the more extensive application of machinery in the elaboration of the resulting iron.

The methods of refining in the open hearth may be classified under three heads, according to the number of times the metal requires lifting or breaking-up, from the period when it is first fused until the bloom has been made ready for placing under the hammer. In accordance with this distinction, the method employed is distinguished as a single, double, or manifold running-down process, or in Germany, where a few small open-hearth establishments are still in operation, as *einmalige, zweimalige*, or *mehrmalige Schmelzerei*. This distinction is, in a great measure, due to the number of heats employed during the operations requisite for the conversion of pig-iron into malleable metal.

In the first, or single method, employed in Styria, white pig-iron, nearly approximating in composition to refined metal, is used, and the elimination of combined carbon is principally effected by the addition of oxidizing agents, without material assistance from the injected blast; the product is a steely iron, from which the excess of carbon is subsequently removed by subjecting the bloom to successive welding heats. In the second, or double process, the metal is run into the hearth or bloomery from a melting-down or running-out fire; and in the last, of which the old German or Walloon forge may be taken as a type, the three operations of converting grey-iron into white, lifting, breaking-up, and the final balling are performed in the same hearth.

GERMAN OR WALLOON FORGE.—The working of the German forge may be described generally as follows, although the various operations admit of more or less modification in accordance with circumstances, and the custom of the district in which the works may be situated.

This operation is carried on in a small open fire, of which fig. 80 represents a vertical section, and fig. 81 a ground plan; the quadrangular hearth, H, is formed of thick cast-iron plates. The depth of the hearth is about 12 inches, and its width from 2 feet to 2 feet 2 inches.

The blast is introduced by the tuyer, *t*, which projects about 4 inches into the hearth, and is so inclined that its axis may intersect the opposite face, somewhat above the line of its junction with the plate forming the bottom.

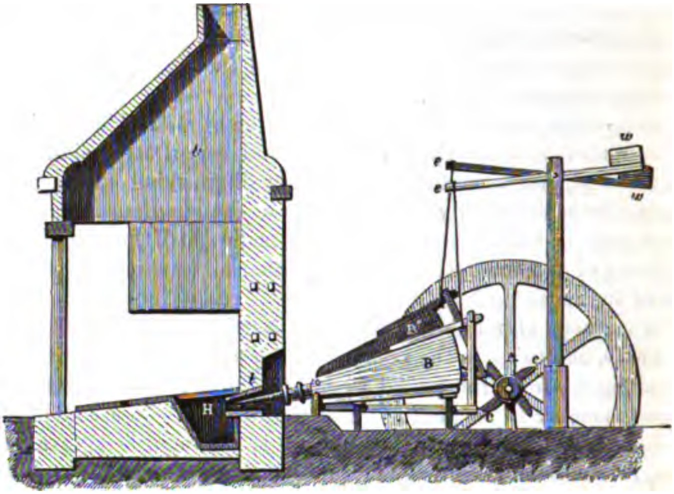

Fig. 80.—German Forge; vertical section.

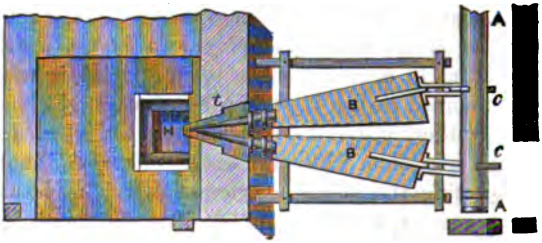

Fig. 81.—German Forge; plan.

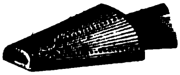

Fig. 82.

The tuyer is made of sheet-copper, and is of the form represented, fig. 82. In this are placed the nozzles of two wooden bellows, B, set in motion by a water-wheel, and arranged so as to afford a continuous stream of air. The movable lids of these are raised by cams, *c*, placed on the axle, A, of the water-wheel, and the too rapid fall of the vibrating segments is checked by their being attached to the levers, *e*, provided with boxes, *w*, in which are placed weights, for the purpose of regulating the rapidity of the descent. The cams, *c*, are so disposed around the axle of the wheel that the movable half of one bellows begins to be raised precisely when that of the other is being released, and in this way a continuous current of air is constantly kept up in the furnace.

In front of the fire-place is a cast-iron plate, raised on one side to the level of the hearth, and on the other inclined to that of the refinery floor. An aperture, called the *chio* or slag-hole, passes through the side of the furnace, and enters the hearth at the bottom; by this aperture the fusible slags are occasionally run off. Over the furnace is placed a hood, *v*, which is made of brickwork, and being provided with a chimney, serves to carry off the smoke and gases evolved during the process. To the sides of this hood are attached plates of sheet-iron, for the purpose of screening the workmen from the heat to which they would otherwise be exposed.

In order to understand the working of this furnace, we must suppose that an operation has just been terminated, and that the hearth still contains a considerable quantity of incandescent charcoal.

The workman begins by filling the hearth with fresh fuel, and then gradually admits the blast. In the older forges the supply of air is regulated by limiting the supply of water on the wheel by which the bellows are set in motion, but in some of those of more recent construction blowing cylinders are employed, and in that case the blast is adjusted by a valve situated near the tuyer.

The iron to be refined is cast either into pigs of from 10 to 15 feet in length, or into short bars or thin plates. In the first case the bar is placed on iron rollers, and its extremity introduced into the middle of the hearth at a height of from 6 to 9 inches above the bottom. When, on the contrary, the metal to be refined has been cast into smaller masses, they are piled to the amount of from 2 to 3 cwts. immediately upon the charcoal with which the cavity of the furnace is filled.

After a short time the metal begins to melt, and passing through the current of air from the tuyer, falls to the bottom of the hearth. This period of fusion ordinarily lasts from three to three and a half hours, and during that time advantage is taken of the heat developed to weld together and form into bars the metal refined during the preceding operation. The drops of melted iron, in passing at a high temperature through the air furnished by the blast, becomes partially oxidized, and by subsequent reaction of the basic silicates of iron formed, a considerable portion of the carbon is consumed.

On arriving at the bottom of the hearth, the iron thus treated has become to a certain extent decarburized, and forms a pasty mass beneath the layer of fuel through which it has passed. The slag, which gradually accumulates in the furnace, is from time to time run off through the tapping-hole before referred to, care being taken to retain a sufficient quantity to carry on the process of decarburization.

The oxidation of the iron is also promoted by bringing the melting mass immediately before the current of air thrown in by the tuyer. The slag run off is preserved for use in the succeeding operation.

When the partially refined bloom has become sufficiently resistant, the workman, by the aid of a strong bar of iron, rolls it up in the form of a ball, and raises it on the top of the fuel, which he now thrusts down into the bottom of the furnace. Fresh charcoal is at the same time added, and the pressure of the blast so increased that the mass is again subjected to strongly oxidizing influences, and a second time reaches the bottom of the hearth, where, from having now lost a considerable portion of its carbon, it forms a spongy mass. The detached fragments are now collected with an iron bar, and united into one mass. Should any portions appear to be imperfectly refined, they are again brought into a position to be directly acted on by the blast.

When the mass has become sufficiently coherent the slag is run off, and the iron is rolled into a large ball, and removed from the hearth by strong levers, and subjected to the action of a heavy hammer, by which the spongy matter is consolidated and welded together, while the siliceous slag is at the same time expressed from its pores. During this operation the hearth is cleaned out, and the larger portion of the remaining slag is drawn off; a certain quantity is nevertheless retained in the furnace to assist in the decarburization of the succeeding charge of cast-iron. Before again proceeding to charge, it is frequently found necessary to cool the hearth with water, which is made to circulate beneath the bottom plate.

The richer slags thus removed are not thrown away, but, together with the scale produced during the hammering of the mass, are employed in the next operation, during the first melting of the pig-iron.

After being removed from the hearth, the bloom is transported to an anvil, on which it receives the repeated blows of a heavy hammer, set in motion by a water-wheel, the slag becoming completely expelled, and the iron consolidated into the form of an elongated prism.

The hammer-head commonly weighs from 800 to 1,200 lbs., and is sometimes made of cast-iron, although wrought-iron is also employed, and in that case it is provided with a face of hardened steel.

The anvil is, in most instances, of cast-iron, which, to give it greater solidity, rests on a heavy mass of the same material, supported by a large wooden pile firmly fixed in the floor. The wooden beam which carries the hammer is strengthened by bands of iron, and is supported by a strong cast-iron ring, provided with trunnions, on which it turns when the head of the hammer is raised. These trunnions are supported by iron bearings fixed in wooden supports. Parallel to the hammer-beam, and at a short distance from it, is situated a horizontal axle, moved by a water-wheel, and provided with a series of cams, which, by coming in contact with an iron band, situated at about one-third part of the distance from the head to the trunnions, forming the centre of suspension, first lift the hammer, and then allow it to fall with its whole weight on the anvil beneath. To accelerate the fall of the hammer when lifted to its full height it comes in contact with a long piece of elastic wood, which acts as a spring, and, by causing the rapid descent of the hammer, prevents the falling beam from coming in contact with the cam which is next in the series. The extreme range of the hammer, or the height to which it is raised from the anvil at each blow, varies from 2 feet to 2 feet 6 inches.

When the working of a piece of iron has been completed, the hammer is propped to the full height of its course by a wooden support, which is removed as soon as the succeeding bloom has, by means of proper tongs, been placed upon the anvil. At first the water-wheel is made to revolve very slowly, and consequently a considerable interval occurs between each blow; but by degrees a more plentiful supply of water is admitted, and the hammer soon attains its maximum speed, which is continued to the end of the operation. Whilst the bloom is being worked on the anvil, it is so turned by the workmen that all its sides successively become exposed to the hammer; and by this means the slag is rapidly expelled from the spongy metal, which is speedily formed into an elongated mass, of which the various parts have become firmly welded together. This is again subdivided, by a cutter, into three or four fragments, which are placed above the bloom formed in the next operation, and when sufficiently heated are drawn into bars, under a hammer specially adapted for that purpose. The mass is divided by a kind of knife, placed on it whilst under the hammer, which, in its fall, strikes the back of the cutter, causing it to divide the iron.

The hammer used for drawing the divided bloom into bars, is, in most instances, much lighter, and makes a greater number of blows in a given time, than that employed for expressing the slag from the bloom when it first comes from the refinery. This hammer, which has less lift than the one just described, is represented by figs. 83, 84. In this case, instead of being raised directly by cams, the motion is communicated on the other side of the centre of suspension; the cam-axle, as in the other hammer, being turned on by a water-wheel. Fig. 83 represents a front view, and fig. 84 a sectional elevation of this hammer. A, represents the axle of the water-wheel, on which are fixed the cams, *c*: these are fitted into a cast-iron ring, which is firmly secured on the shaft by the wedges, *a*, made of hard wood.

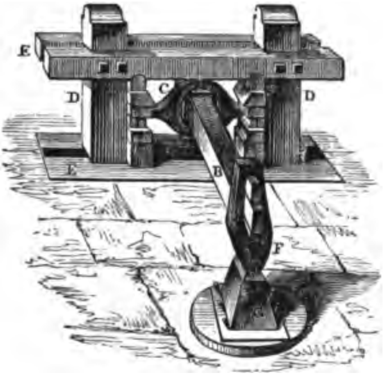

Fig. 83.—Tilt-Hammer; front view.

The beam, B, carries the hammer, F, and is received into an iron ring, C, which is provided with trunnions, working in bearings between the perpendicular posts, D, and the cross-bars, E, which are strongly bolted together.

At the extremity of the beam opposite to that which carries the hammer, is an iron plate, *p*, firmly secured by means of the band, *d*; against this plate the cams, which move in the direction indicated by the arrow, are successively brought to bear, and by their pressure raise the hammer fixed on the other end of the wooden beam, which again falls as soon as the cam in contact with the plate, *p*, has so far depressed the end of the lever as to allow of its passing round without further impediment. A spring, R, is placed under the tail of the beam, instead of above the head. The faces of the hammer, F, and of the anvil, G, which rests on the bed, H, are inclined at a certain angle with the floor; the guide, I, serves to steady long iron bars when worked under the hammer.

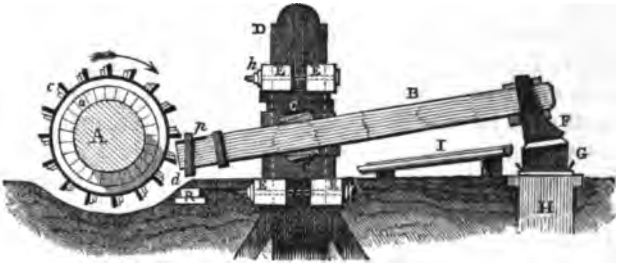

Fig. 84.—Tilt-Hammer; sectional elevation.

The weight of each bloom is in most instances between 1 and 2 cwts., and 100 lbs. of cast-iron produce about 85 lbs. of bars. For every 100 lbs. of wrought-iron obtained, 150 lbs. of charcoal are employed. The bellows are stopped as soon as the bloom is ready to be placed under the hammer, and the whole operation occupies about five hours. The iron manufactured by this method is of excellent quality. Refineries of this description are sometimes supplied with heated air instead of the ordinary cold-blast, and attempts have been made to replace the employment of charcoal by the use of coke; but the iron produced by this means is so much inferior to that prepared with the usual fuel as to more than compensate for the advantages derived by the substitution of the cheaper combustible.

The process above described is called by the Germans *Klumpfrischen*, or lump-refining, and differs from the *Durchbrechfrischen*, because in the latter the bloom, instead of being rounded together in one mass on the hearth of the furnace, is then separated into several pieces, which are successively worked under the hammer.

The French call the first process *affinage au petit foyer*, and the second, *affinage par portions*.

A further distinction between the open-hearth processes may be founded upon the various methods adopted for working the iron, as it loses its carbon and becomes converted into malleable metal, or, as that transformation is called in this country, it *comes to nature*. When of good quality the whole charge may be lifted together, and worked in a single mass before the tuyer, whereas with inferior descriptions the particles of iron, instead of being allowed to coalesce as they form, are divided into several portions, which, after being separately decarburised, may either be worked into one ball or forged separately.

Three principal methods of manufacturing iron in the charcoal hearth are employed in Sweden:—the Walloon, the Franche-Comté, and the Lancashire processes. The first of these is confined to forges producing Dannemora steel-iron. The Franche-Comté and Lancashire hearths are covered and provided with flues in which the charge of pig-iron is heated previous to fusion, and where the blast is raised to a temperature of 100° C. by being passed through a series of cast-iron pipes; the pressure employed is from 1 lb. to $1\frac{1}{4}$ lb. per square inch.

In the first method the bloom, after shingling, is cut into two pieces and re-heated in the same fire; while in the Lancashire forge, either a second hearth, or a gas welding-furnace is employed for re-heating. In both cases the weight of bar-iron produced is about 85 per cent. of that of the pig-metal operated on, and, under the most favourable conditions, the consumption of charcoal is about one and a half times the weight of the finished bars made.

IRON FOR TIN-PLATES.—In South Wales a superior description of iron employed for rolling into the thin sheets formerly much used in the manufacture of tin-plates, was made in the charcoal hearth. The pig-metal treated was of good quality, and was smelted either with anthracite or in a coke furnace blown with cold-blast. The charge, which consisted of from 5 to 6 cwts. of good Welsh mine or hæmatite pig, was first melted in a small coke refinery with two tuyers, and, after the necessary amount of exposure to the oxidizing influences of the blast, was run off, through an inclined gutter, into charcoal hearths placed in front, and at a lower level. These hearths were made of cast-iron plates, their bottoms being cast hollow in order that they might be kept cool by the circulation of a current of cold air; the charge of refined metal was divided equally between them, and each blown by a single tuyer. Water-tuyers were made use of both in the refinery and charcoal hearths, although cold-blast only was employed in each case. In the charcoal hearth the metal was frequently broken up and raised by an iron bar, the slags being run off two or three times during each operation, of which the average duration was little more than an hour. The whole of the charge was worked into a single ball, which weighed something less than 2 cwts., and was first shingled under a tilt-hammer and then drawn into a long flattened bar about 2 inches in thickness. This was partially cut through transversely, and broken by the blows of a sledge-hammer into fragments called *stamps*, each weighing about 28 lbs. The fracture of the pieces thus broken off was examined, and such only as presented a finely crystalline and uniform grain were used in the formation of the pile from which the finished sheet was made.

The re-heating of the stamps was conducted in a special furnace known as the *hollow fire*, consisting of a deep rectangular hearth roofed over at top. This hearth was partially filled with coke, and supplied with blast through a single inclined tuyer introduced near the top of the fuel.

The piles, consisting of the fragments detached from the rough bars by the process above described, were supported upon a *staff* or flattened bar of iron, above the top of the fuel, but fully exposed to the action of the flame from the hearth. At the present time mild steel has almost entirely superseded the use of wrought-iron in the manufacture of tin-plates.

In the West Riding of Yorkshire the same method of breaking and selecting the rough bars is employed in those forges which are most noted for the superiority of the iron they produce.

## PREPARATION OF MALLEABLE IRON BY THE REVERBERATORY PROCESS.

PUDDLING.—The reverberatory process for converting cast into malleable iron, introduced by Henry Cort, in 1784, has now almost entirely superseded all others in localities in which mineral fuel is moderately abundant; while the introduction of gas furnaces has been the means of its becoming very general, even in wooded districts and in localities where inferior descriptions of fuel only are available. Not only wood, but also brown coal and peat, may, when converted into carbonic oxide, be employed as fuel in the puddling-furnace. The reactions which take place during the operation of puddling are of a similar character to those of the open hearth, as the decarburization of pig-iron is effected by the joint action of a current of atmospheric air and oxidizing fluxes. In the puddling-furnace, however, a current of air is obtained by the draught of a high chimney instead of by a blast, and according to the relative importance of the effect produced by the action of atmospheric air or by oxidizing fluxes respectively, the process is spoken of as the "dry" or the "wet" method of puddling. In the old process of *dry puddling* the necessary oxygen is chiefly derived from atmospheric air, while on the contrary, in *wet puddling*, or *pig-boiling*, it is, to a very large extent, furnished by the slags and oxides of iron added. In either case the conversion of grey-pig into white-metal, by a preliminary fusion, is advantageous, and the operation thereby accelerated.

The fire-place of the modern puddling-furnace, figs. 85, 86, and 87, (slightly modified from Percy) is rectangular and divided from the hearth by a low fire-bridge; the roof, which is a flat arch, has a gradual slope towards the flue. The fire-bars, *a*, are movable for greater convenience in removing the clinker, and a powerful draught is obtained by means of a brick chimney, *b*, from 30 to 50 feet in height, strengthened by iron ties; at top this shaft is furnished with a sheet-iron damper, *c*, opened and shut by a lever and chain, by which the workmen can regulate at will the amount of air passing through the furnace. The outside walls are enclosed with strong side-plates of cast-iron, united by flanges and bolts, and are bound together by wrought-iron tie-rods across the top. By this means not only is the perfect solidity of the structure insured, but the entrance of air through rents in the brickwork entirely prevented. The bottom of the bed, *d*, is formed of cast-iron plates united by tenon-joints, and supported on dwarf pillars of the same metal. The sides of this bed are variously constructed, according to the method of artificial cooling adopted, but they are frequently made of hollow iron castings, through which a current of air circulates for the purpose of protecting them from the intense heat of the furnace. The hearth, 6 feet in length,* is terminated at either end by a straight wall of fire-brick, that nearest the fire-place being called the *fire-bridge*, and the other, at the opposite end, the *flue-bridge*. The brickwork is made to overlap the top of the side-frame in such a way as to form a recess for the refractory material

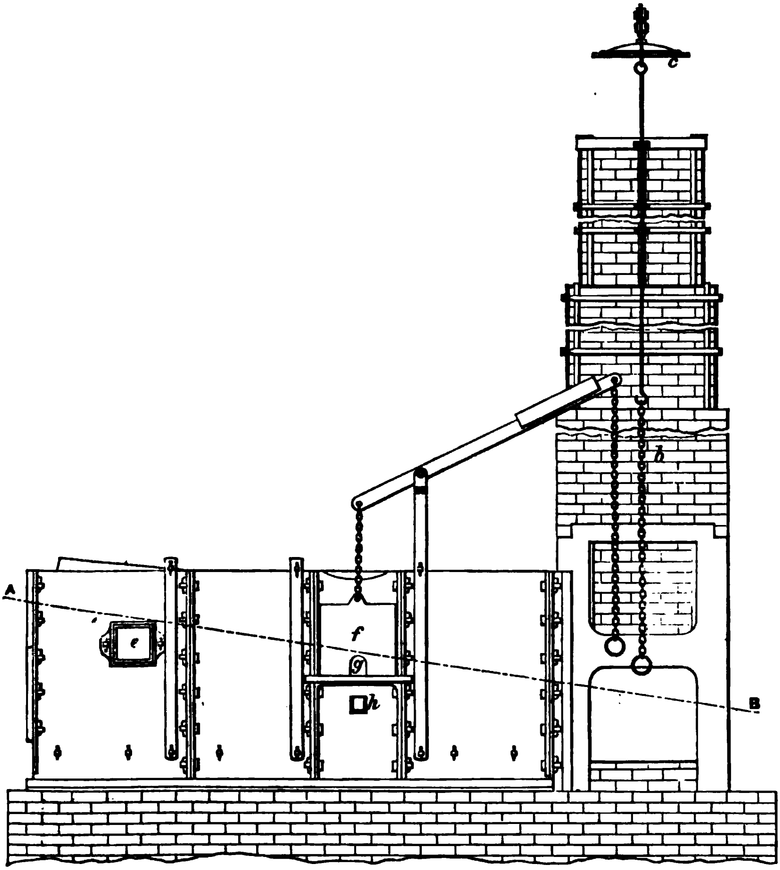

Fig. 85.—Puddling-Furnace; side elevation.

(*fettling*) with which it is lined. The width of the hearth is 3 feet 9 inches at one end, and 2 feet 9 inches at the other.

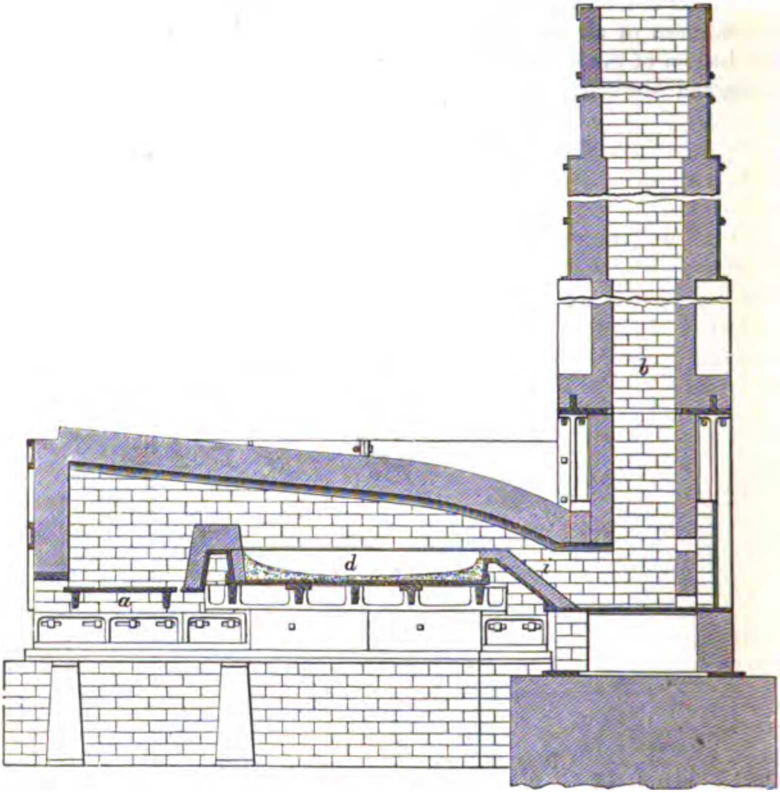

Fig. 86.—Puddling-Furnace; section on C, D (fig. 87).

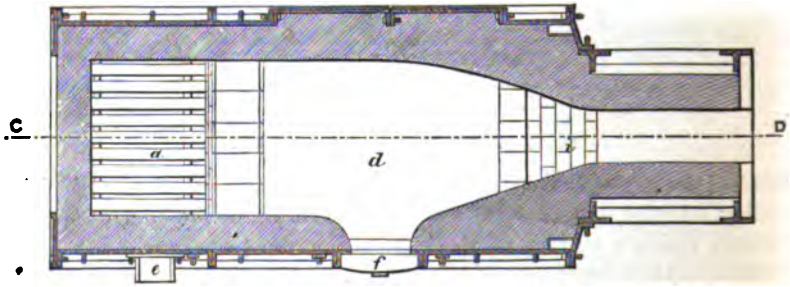

Fig. 87.—Puddling-Furnace; section on A, B (fig. 85).

The depth of the fire-place varies with the nature of the fuel employed, being greatest when the coals used are but slightly bituminous. For furnaces provided with an ordinary fire-grate, the best fuel is non-caking coal, containing but a small amount of sulphur, and burning with a long flame. The grate-area should be between one-half and one-third that of the bed; or, taking the latter at 20 square feet, the area of the grate will be from 7 to 8 square feet. The firing-hole, *e*, which is placed about a foot above the grate, has no door, but is closed by throwing a shovelful of coal on a projecting ledge cast upon the iron casing, and piling it against the opening. The flue—of which the sectional area varies with the nature of the fuel employed, being for bituminous coal one-fifth that of the grate, and for anthracite about one-seventh—slopes gradually towards the stack. In some cases, a second bed is placed behind the flue-bridge, on which the pig-iron destined for the succeeding charge is heated by means of the flame which passes over it on its way to the chimney; in others when, as in some gas furnaces, a blast is used, the air is heated by first passing through the hollow side-frames of the hearth, and afterwards through a coil of iron pipes situated between the furnace and the base of the chimney. The walls of the stack are of common brick, with an internal lining of fire-brick, and the working-door, *f*, which is situated on the same side of the furnace as the fire-hole, is a large fire-tile, set in a cast-iron frame, suspended by a chain to a lever having a counterbalance-weight at the other end. This is only opened for the purpose of charging, or for removing the puddled balls, but a small rectangular or arched notch, *g*, called the *stopper-hole*, is cut out of its lower edge for the introduction of the tools used in stirring or rabbling the charge. The sill of the working-door is about 10 inches above the bottom of the bed of the furnace, and below it is a tap-hole, *h*, usually kept closed, through which the slag, or *tap-cinder*, is withdrawn from time to time as may be required. Another portion of the slag overflows the flue-bridge, and runs down the inclined flue to the bottom of the stack, where it is allowed to accumulate.

The portion of the bed opposite the working-door is of a curved form, and in the ordinary single furnace is only accessible from one side; but double furnaces have doors on both sides, so that two sets of puddlers may work at the same time on a charge, of which the weight is proportionately increased.

The working bed, or lining, of the puddling-furnace was formerly composed of sand, but substances rich in oxide of iron are now employed. These not only materially assist the process, but also, under certain circumstances, improve the quality of the metal produced. The cast-iron bottom is usually prepared for use by being covered with a layer of tap-cinder or hammer-slag, which is heated until it has assumed a pasty condition and is then worked down and uniformly spread over its surface; this is covered by a thin layer of fettling composed of nearly pure oxide of iron. The thickness of the finished coating need not exceed 1½ inch, and the first charge should consist of scrap-iron, or of waste blooms and refined metal. Grey-pig should not be puddled alone before the refractory lining has become sufficiently consolidated to resist the action of silica resulting from the oxidation of silicon contained in the metal.

The side-plates of the hearth as well as the bottom require to be lined with some substance rich in oxide of iron; the best materials for this are the pure oxides of iron, such as hæmatite, magnetite, or roasted spathose ore, free from earthy matter. Roasted tap-cinder, known as *bull-dog*, is extensively used, as is also *purple ore*, or *blue billy*, which is the residue from the treatment of roasted cupriferous Spanish and Portuguese pyrites by the wet process for extracting copper.[1]

Other substances are occasionally employed for fettling, but in selecting materials for this purpose all such as contain a notable quantity of quartz should be carefully avoided. The presence of a certain amount of clay is only injurious as diminishing, in some degree, the durability of the lining. The most desirable material for fettling is what is known as *best tap*, which is the cinder from a re-heating furnace in which piles of wrought-iron are prepared for rolling on a cast-iron bottom, known as a cinder-bottom, in contradistinction to the original brick hearth covered with sand. A furnace fettled wholly with best tap so affects the quality of the metal produced, that good iron may be made even from cinder-pig, while from pig of fair quality the weight of puddle-bar produced is greater than that of the pig-iron operated upon. Titanic iron ore, or ilmenite, forms a very durable fettling, but is said to render the iron cold-short. The process of refining iron in the old refinery is still carried on in some works, but has, to a large extent, been superseded by pig-boiling, or puddling pig-metal without previous preparation.

The process of puddling may be described as including four distinct operations, namely, the melting-down of the charge; its incorporation at a low heat with oxidizing fluxes; the elimination of carbon by stirring, with exposure to air at a high temperature; and, finally, the preparation of balls of spongy metal suitable for squeezing or hammering. Although susceptible of considerable modification, the following may be considered as a general outline of the ordinary method of puddling.

As soon as the charge has been introduced into the previously heated

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[1] The average composition of this substance is as follows:

| | |
|---|---|
| $Fe_2O_3$ | 96·00 |
| Pb (as sulphate) | 0·75 |
| Cu | 0·20 |
| S | 0·36 |
| P | none |
| Ca | 0·40 |
| Na | 0·10 |
| Co, As and Cl | traces |
| Insoluble residue, &c. | 2·11 |
| | 99·92 |
| Metallic iron | 67·00 |

furnace, the working-door is closed, and the joints, if necessary, luted with clay; the grate is also pricked, fresh fuel added, and the firing-hole stopped with slack, in order that no air may enter except through the ash-pit. At the expiration of about a quarter of an hour, when the metal begins to soften, the puddler introduces a bar or *rabble* through the notch in the bottom of the working-door, and removes any unmelted lumps from the sides of the furnace to the middle of the hearth. The fire is now increased during a few minutes, and as soon as the metal has become uniformly liquid it is briskly stirred, the temperature being at the same time gradually lowered by partially closing the damper on the top of the stack, until the surface of the charge has, by the formation of a covering of slag, become protected from further oxidation by the action of the air passing through the furnace. The management of the operation immediately after charging is more or less varied in accordance with the nature of the metal treated. With grey-iron, which becomes exceedingly liquid when fused, the fragments may, when the furnace is sufficiently hot, be distributed equally over the bed; when, however, it is less highly heated, the pigs of metal are piled near the fire-bridge, and as the operation proceeds and the heat increases, the unmelted portions are drawn into the centre, and forced beneath the surface of the fused slag. When, on the contrary, white or refined iron is operated upon, it is advantageous to bring the furnace to a high heat before the introduction of the charge. By this means the metal is made to fuse more rapidly, and is less subject to oxidation than when the operation is more prolonged.

In order to obtain the full benefit of the oxidizing power of the slags, it is necessary that they should be intimately incorporated with the metal, and for this purpose the draught is checked, the temperature lowered, and the charge, while in a pasty state, well stirred. Hammer-slag or mill-cinder is also added for the purpose of rendering the slags more basic, and to compensate for the silica resulting from the oxidation of the silicon of the pig-iron. As soon as the mass has thus become pasty the reaction of the oxides and silicates of iron upon the carbon of the metal becomes apparent, and copious blue flames, resulting from the combustion of carbonic oxide, make their appearance. The damper is now opened, and on the temperature becoming higher, the surface begins to boil from the rapid escape of carbonic oxide; some of the slag is run off, and the action is accelerated by stirring with a hooked iron bar or rabble. At this point the puddler, using the side of the door-frame as a fulcrum, sweeps every portion of the bed from the centre towards the bridges, and in order to prevent the tool from becoming too hot and adhering to the metal, it requires to be frequently changed. On being taken out of the furnace it is cooled in a cistern or *water-bosh*, by which the adhering cinder becomes detached, and the hook at its end is afterwards hammered into shape. In proportion as the carbon becomes eliminated, the violence of the ebullition is diminished, the mass begins to stiffen, or "come to nature," and malleable iron begins to make its appearance. To prevent a too rapid agglomeration of the charge, the contents of the furnace are again broken up, and thoroughly mixed by stirring; any pasty lumps adhering to the sides are detached, and the mass is subjected to a final heat for the purpose of rendering the cinder perfectly liquid, and thereby facilitating its separation from the metal.

The last operation consists in forming the metal into balls; this is done by detaching from the charge masses each weighing from 60 to 80 lbs., and compressing them with the tool until they have acquired sufficient coherence to admit of being moved without falling to pieces. This may be effected either by pressure against the bottom and sides of the furnace, or by so rolling a small nucleus of metal on the hearth as to collect other fragments which become attached to it by welding. As soon as a ball has been thus prepared, the workman, by means of a strong iron tool, places it close to the fire-bridge on the far side of the furnace, for the purpose of protecting it against the action of the air entering the working-door and passing off by the chimney; the making of the second ball is then at once proceeded with in like manner, and when the whole charge has been balled up, the working-door and stopper-hole are closed and the last heat given.

Finally, the balls are drawn, one by one, to the working-door, lifted by suitable tongs to the iron table in front of it, and afterwards, either dragged along the floor, or, more frequently, carried on a small wrought-iron truck to the machine by which the *shingling* or first compression of the metal is effected.

The old system of dry puddling is applicable to the treatment of white or refined metal only, and, as before stated, the oxidation of the carbon is more dependent on the action of atmospheric air than it is in pig-boiling; the quantity of slag produced is also considerably less. By this process, as soon as the metal has been melted down and has assumed a pasty state, it is broken up and kept constantly stirred for the purpose of incorporating with it the oxide produced during the operation. The charge of the furnace is maintained in a partially fused or pasty state, and the stirring goes on almost continuously from the running-down to the balling-up. As, however, the use of sand bottoms is attended with great loss of iron, and the metal produced is of inferior quality, they have, at the present time, become obsolete.

The charge of a puddling-furnace is, in Staffordshire, from 4 to 4½ cwts., and from five to seven heats are worked off by a puddler and his assistant during a *turn* of twelve hours; the difference of weight between the pig-iron charged and the puddled bars obtained is from 7 to 10 per cent. The coal consumed is about equal in weight to the puddled bars made, and the fettling materials required, per turn, are from 6 to 7 cwts. of bull-dog, and 2 to 3 cwts. of puddler's ore, or blue billy, to which must be added the mill-scale introduced into the charges. In Scotland, where dark-grey metal, containing a large amount of silicon, is puddled without being previously refined, from four to five heats only, each of 4 cwts., are made in twelve hours; in this case the loss experienced, from pig-iron to puddled bars, is from 15 to 18 per cent., and the consumption of coal per ton of the latter varies from 25 to 26 cwts. In Cleveland the consumption of small coal is from 24 to 27 cwts. per ton of puddled bars made.

Wrought-iron of very superior quality is manufactured in the West Riding of Yorkshire from cold-blast refined metal. The furnace employed is of comparatively small size, and is provided with a very high stack, for the purpose of insuring a strong draught. The charge, weighing 3 cwts., is heated to redness before its introduction into the furnace, and the melting-down is effected in from twenty to twenty-five minutes; the whole operation occupies about one hour and twenty minutes, and nine heats are made in the course of twelve hours. Three or four balls only are obtained from each heat, which, after shingling under a helve-hammer, are made into stamps from 10 to 12 inches square and 2½ inches in thickness; these are broken by the fall of a heavy weight, and subsequently assorted in accordance with the nature of the fracture which they severally exhibit. Those which are most uniformly crystalline are employed for the manufacture of hard bars, while those showing a distinct fibre are reserved for making into boiler-plates and wire-rods. The consumption of coal is about 30 cwts. per ton of fine metal treated. In Belgium the coal consumed is usually equal in weight to the puddled bars obtained, and the loss on cast-iron is from 7 to 10 per cent., according to the quality of the metal operated on.

*Puddling in Gas Furnaces.*—In Carinthia gas puddling-furnaces are employed, the fuel being air-dried wood, which is converted into gas in a generator having a capacity of about 14 cubic feet; air is introduced near the bottom at a pressure of half an inch of mercury, and the combustion of the gases is effected by a second blast brought in immediately above the fire-bridge, through an oblong tuyer extending the whole width of the hearth. This blast is heated to a temperature of 200° C., by passing through the hollow side-plates of the furnace, and a second bed, situated between the puddling-furnace and the chimney, is used for heating up the metal which is to constitute the next charge. In Styria, lignite is sometimes employed for puddling, the consumption being from 22 to 24 cwts. per ton of blooms, and the loss of weight on the metal from 6 to 10 per cent. From 200 to 280 cubic feet of wood are required to produce 1 ton of blooms, and from 240 to 360 cubic feet of turf are capable of yielding the same result; the consumption does not appear to differ materially, whether it be consumed on an ordinary grate or converted into gas. A gas puddling-furnace employed at Neustadt (Hanover), in which turf is the fuel used, is represented, figs. 88, 89, of which the first is a longitudinal, and the second a horizontal section above the level of the hearth. The gas-generator, A, is supplied with fuel by the hopper, *a*, and at bottom is provided with a set of fire-bars, *b*; the air necessary for the conversion of the carbon of the turf into carbonic oxide gas is supplied from the blast-main of the establishment, through the tap, *c*, passing into the ash-pit beneath the bars. Another portion of the blast enters the cast-iron sides, *d*, of the hearth through the pipes, *e*, whence, after becoming to a certain extent heated,

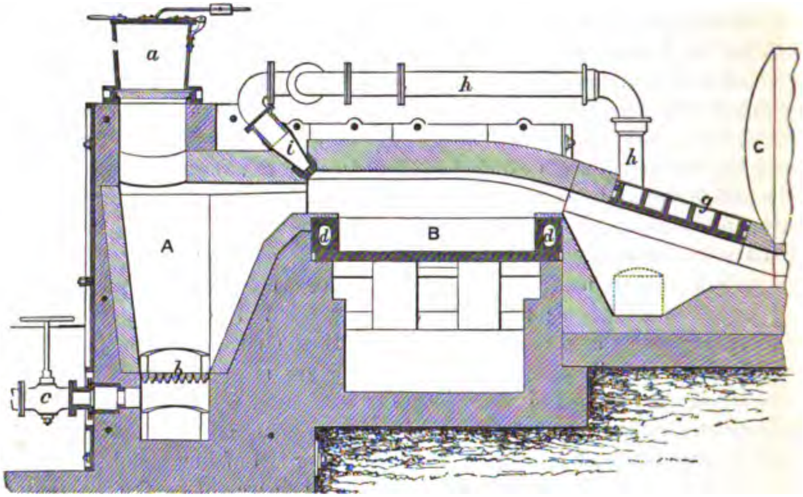

Fig. 88.—Puddling-Furnace, Neustadt; longitudinal section.

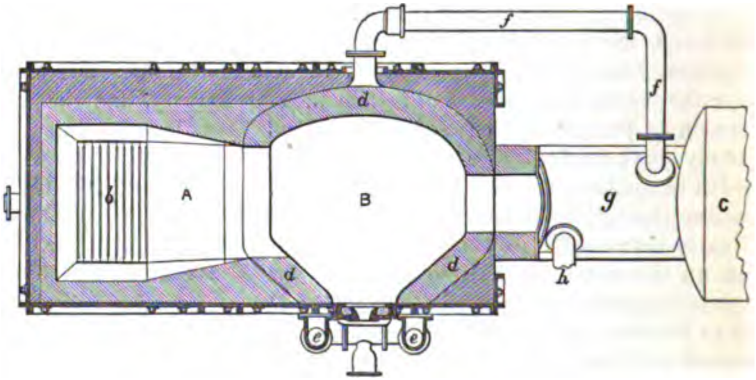

Fig. 89.—Puddling-Furnace, Neustadt; horizontal section.

it passes off by the pipe, *f*, to the heater, *g*, which covers the flue of the furnace. This is made of cast-iron with a sheet-iron top, and has a number of divisions cast on the lower plate, through which the blast circulates in a zigzag direction. Here the air, which has become heated by passing through the sides of the hearth, is further elevated in temperature, and escaping by the pipe, *h*, is conducted to the inclined tuyer, *i*, which is oblong in form and extends completely across the bridge. The bottom and side-plates of the hearth, B, are shown in the drawing without any lining, but this requires to be added before the furnace can be used for puddling; the heated gases escaping from the apparatus are finally utilized by passing under and around the steam-boiler, C.

Among the various contrivances employed for puddling cast-iron, Siemens' regenerative gas-furnace is of great importance. The way in which this principle is applied is shown, figs. 25, 26, 27, which represent a gas re-heating furnace on this plan. The difference in general form between a re-heating and a puddling-furnace is not very great, and consequently it will be easy, in consulting drawings of the one, to understand the application of the system to the other.

Two men, a puddler and his assistant, usually conduct the working of each furnace; the former does the heaviest portion of the work, including making up the balls, while the latter attends to the firing, and also does a portion of the stirring and rabbling.

The tools employed in puddling are of two kinds, namely, long straight chisel-shaped bars or *paddles*, and hooked flat-ended bars or *rabbles*. When withdrawn from the furnace, the ends of these are coated with molten cinder, which is removed by quenching in a cistern of cold water.

Various attempts have been made to puddle iron by machinery, and the methods proposed for effecting this object may be classified under two distinct heads, namely, by imitating the motions of hand-stirring by mechanical appliances, and by using rotatory or oscillatory hearths.

*Mechanical Rabbles.*—For some time mechanical rabbles have been attached to puddling-furnaces for the purpose of diminishing the heavy labour of stirring; they are, however, useless in the laborious operation of balling, and their adoption has not become general.

In Eastwood's mechanical stirrer, which is one of the simplest machines of this class, the rabbling-tool is suspended in a stirrup at the end of a bent lever, which receives a reciprocating motion by means of a crank. The centre of oscillation of this lever is at the extremity of an inclined jib, to which a lateral motion is imparted by a rod working on a pin attached to a screw-wheel, driven by a worm on the crank-shaft. Motion is given to this arrangement by a chain passing over a pulley, and the rabble is moved backwards and forwards across the hearth, once in each revolution, while at the same time its point of suspension is shifted a short distance by the movement of the jib, caused by the screw-gearing. In this way a compound motion is communicated to the tool, which causes it to travel gradually over every portion of the furnace bottom. This apparatus is bolted to the casing-plates of the furnace, and the driving-pulley is connected to the shaft by a fast-and-loose clutch. The furnaces of Tooth, Menelaus, and Danks are themselves movable.

*Rotative Furnaces.*—The first experiment with a rotative furnace attended with any degree of success was that of Tooth, who used a wrought-iron cylinder lined with fire-brick; this was made to revolve between a fire-place and the flue leading to a chimney; the flames passed through the cylinder, and the balls, when ready, were withdrawn from the end. It was found that the brick lining of this machine rapidly wore away, and that the iron was imperfectly puddled, as the mass obtained frequently enclosed a central lump of unchanged cast-iron.

Menelaus improved upon Tooth's machine, and the results obtained were much more satisfactory. The rotating portion was egg-shaped inside; the idea being that whereas in Tooth's machine the iron was simply rolled round, in the elliptical revolver, from the different diameters at various parts, the action would be to so break up the iron as not to form a cylindrical mass like that produced in Tooth's machine; thus in turn exposing every portion to the oxidizing action of the air and cinder. This revolver was removable, and by means of a crane and a pair of trunnions was moved from the fire-place, tilted on end, and the lump of iron turned out, in one mass, upon a bogie, and taken to the hammer. The experiments of Menelaus appear to have, practically, failed to give satisfactory results solely on account of the difficulty experienced in finding a suitable lining; he first tried ganister and various descriptions of brick and ilmenite. Ilmenite was found to stand best, but it was difficult to fix it in the furnace, and it moreover made the iron cold-short.

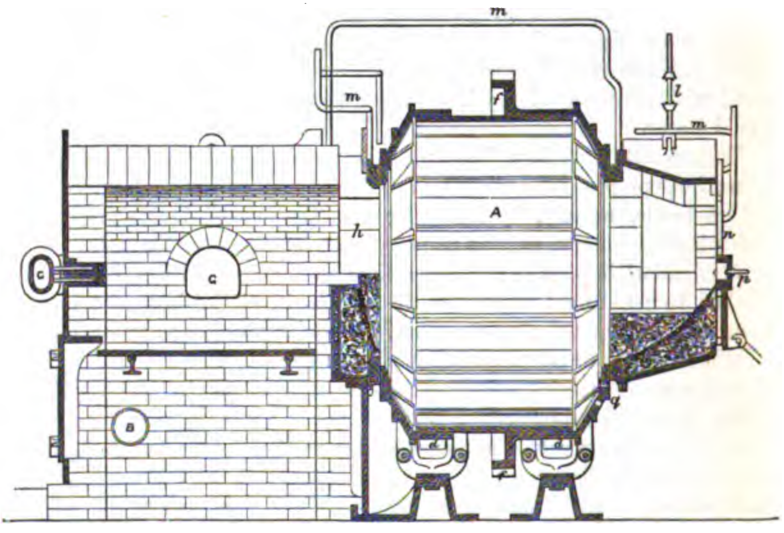

Fig. 90.—Danks' Puddling-Machine; longitudinal section.

Danks' rotative puddling-furnace, which was invented in America, and subsequently introduced into this country upon a large scale after being favourably reported upon by a Commission of the Iron and Steel Institute, is represented in longitudinal section, fig. 90, and in end elevation, partly section through the revolving chamber, fig. 91; fig. 92 is an elevation of the movable end-piece and flue. The revolving cylindrical

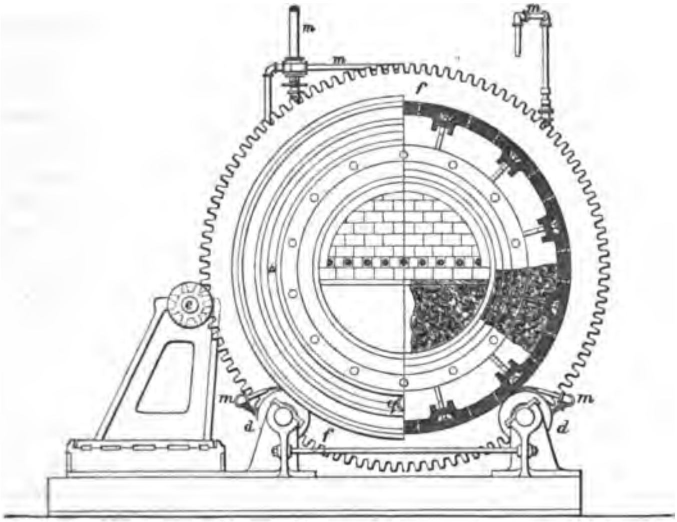

Fig. 91.—Danks' Puddling-Machine; end elevation, partly section.

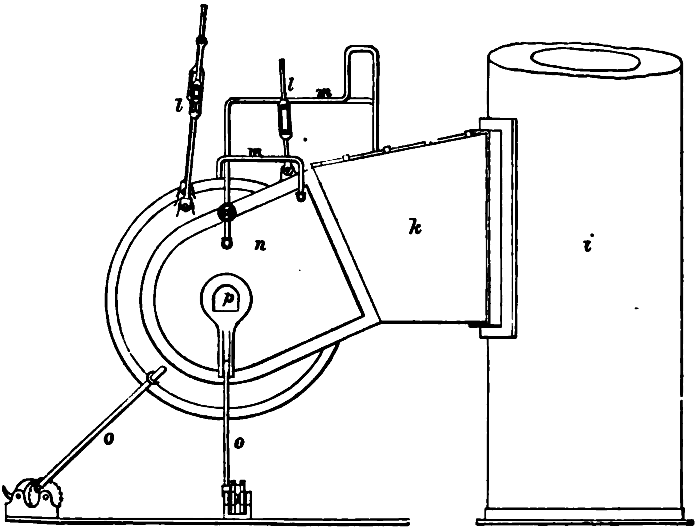

Fig. 92.—Danks' Puddling-Machine; end-piece.

chamber, A, is made with wedge-shaped recesses, which act mechanically in retaining the initial lining in its place. This first lining may be composed of any iron ore, free from silica, ground with cream of lime; this is introduced in the state of mortar, and when dry becomes a refractory and sufficiently coherent material to allow of the fettling being melted upon it without either melting itself or breaking away from the plates. The ore employed for mixing with the lime should, by preference, be anhydrous, since otherwise the removal of its water of combination, by the heat to which it is exposed, is liable to cause the crumbling of the mass.

Upon this foundation a quantity of any ore, free from silica, may be melted, and for this purpose the hydrated varieties may be employed, as their combined water is rapidly driven off. Into the bath of melted ore thus obtained, large solid lumps of the same material are thrown, and these, being cold, cause the melted ore to set round them, and by firmly fixing them in their places a rough internal lining is produced. It is not only necessary that these lumps should be moderately free from silica, but also that their texture should be such as to prevent their crumbling by heat; this was found to be the case with ilmenite. Best tap-cinder answers this purpose very well, and, when suitable ores cannot be obtained, oxidized scrap-iron may be substituted with advantage. The Danks furnace has a closed ash-pit, the necessary air being supplied by a fan-blast through the pipe, B; jets of air are also introduced over the fuel by the nozzles, *c*, in connection with the air-main, C; these, which extend the whole width of the bridge, enable the puddler in charge of the machine to regulate the draught, and he has thus complete control over the furnace.

As time is lost, and a considerable amount of fuel expended in melting the charge in the revolving furnace, the pig should either be fused in a cupola or run directly from the blast-furnace in which it is produced. A jet of water is directed against the lining on the descending side, for the purpose of chilling a portion of the liquid cinder and causing it to be carried under the metal. Mr. Snelus is of opinion that this has also the effect of carrying off sulphur from the cinder, as in the case of Parry's steam-refinery.

When grey pig-iron is treated, the boil commences in about ten minutes, but with white-pig it begins in two or three minutes only after melting. After tapping off the cinder, the cylinder is set in motion and the fire urged; the iron now begins to boil violently, and carbon becomes rapidly oxidized. But little cinder is produced during this portion of the operation, and the greater part of it is removed with the ball. An ingenious arrangement, consisting of a movable end-piece, fig. 92, at the back of the flue, permits the ball to be removed from the Danks furnace by a fork worked by a crane, by which it is placed on a bogie and carried to a hammer or squeezer. The revolving cylinder, A, is supported on friction rollers, *d*, and is set in motion by the pinion, *e*, working in the toothed segments, *f*. G is the firing-hole, and *h* the passage for the flame and gases into the cylinder; *i* is the chimney-stack, *k* the stationary flue, *l* suspension-rods with swivels, *m* water-pipes, *n* water-front or movable piece, *o* supports, *p* stopper-hole, *q* tapping-hole. The firing-hole, G, has a coil of wrought-iron pipe cast in its lining for the purpose of allowing the circulation of a stream of water to keep it cool, and the bridge-plate between the fire and the charge of metal has also a coil of water-pipe cast into it for the same purpose.

In Danks' machine the puddling is entirely effected by the fettling, the carbon, silicon, and phosphorus being almost completely oxidized by it, and by the cinder introduced. The separation of silicon, sulphur, and phosphorus, by this apparatus, is said to be more perfect than by hand-puddling, and the inventor alleges that the more silicon and phosphorus the pig contains the better will be the quality of the iron produced. Strange as this statement may appear, it is confirmed by Riley, who says that the views of Mr. Danks seemed to be borne out by the results of the working of the machine in this country.

The weight of the ball being about 650 lbs., which is larger than could be conveniently handled in the ordinary squeezer, Mr. Danks designed a special machine for shingling, which will be described when treating of forge-machinery. Iron thus produced, after being thoroughly squeezed and re-heated, can be rolled out at once into a rail or large bar, without the operation, now required, of rolling into puddle-bar, piling, re-heating, and again rolling.

The Danks furnace, as orginally constructed, has been found incapable of sustaining the wear and tear of continuous working for any lengthened period, and has therefore been nearly, if not entirely, abandoned, both in Europe and America. At Creusot it is used in an improved form, having the revolving portion surrounded by a water-jacket, which cools the cylinder and preserves its shape. There is also a projecting central bridge likewise cooled by water, which divides the inside into two parts. The movable flue-piece is hinged, and can be moved in or out of the working portion by a rack-and-pinion gearing similar to that of the Bessemer converter. The metal, charged in the melted state in quantities of 20 cwts. at a time, is brought to the balling condition in about half an hour, with a loss of about 15 per cent., giving a mass of puddled iron weighing 17 cwts., divided into two parts by the bridge, which are shingled separately. The consumption of fuel is from 9 to 10 cwts. per ton of blooms, exclusive of that required for melting the pig-metal. The lining requires to be completely renewed after working ten or twelve charges, or about once in twenty-four hours.

A revolving puddling-machine has also been invented by Mr. Spencer, of the West Hartlepool Iron-Works. His converter is rhomboidal in form, having at its two opposite ends axes at right angles to the extremities or discs. These are made to revolve on rollers by suitable gearing. The sides are honeycombed to retain the fettling, consisting of best tap, which is introduced into the recesses, and fresh tap melted over it. Experiments made with this apparatus have shown that the silicon and phosphorus are almost completely removed by it from Cleveland pig-iron containing above 2 per cent. of the latter element.

FORGE-MACHINERY AND OPERATIONS.—That portion of an iron-works in which puddled blooms or rough bars are produced, including puddling-furnaces, shingling-machines, and puddling-rolls, is called the *forge;* while those portions of the establishment in which rough bars are re-heated and transformed into finished or merchant iron, are known as the *mill.* This department comprehends the re-heating or balling-furnaces, together with the mills and other appliances employed in the production of bars, plates, sheets, or other merchantable forms.

*Hammers.*—The compression of the rough balls of malleable iron into blooms is effected by the use of either hammers or squeezers, the former acting by percussion and the latter by compression. In the puddling-forge, the blooms thus obtained are also converted into rough bars by passing them, at the same heat, through a rolling-mill. Hammers of two kinds only were formerly employed in the preparation of blooms; in the first, or tilt-hammer, the axis is placed between the point where the cam acts upon the shaft and the head, while in lifting-hammers, or helves, the hammer-block and lifting-cam are both on the same side of the fulcrum. Both are, however, now very generally superseded by the steam-hammer.

Tilt-hammers are usually small in size, and are driven at a considerable speed, being used rather in drawing out bars and finishing work generally, than for shingling blooms. The shaft is made of one or more beams of straight-grained timber, which, in the latter case, are hooped together with wrought-iron rings, and the pivots either pass through the shaft or are more frequently attached as trunnions to a strong broad hoop. The head has usually somewhat the form of a sledge-hammer; the general arrangement of a hammer of this description is shown in figs. 83, 84, pp. 285–6.

Helve-hammers are of two kinds: the nose, or frontal helve, in which the cam acts upon a projection immediately in front of the hammer-block; and the belly helve, which has the cam-shaft below the level of the floor, and which acts upon it about midway between the fulcrum and the head. Hammers of this description, such as were formerly in general use in puddling-forges, have been made of all weights up to 10 tons; the most usual sizes are from 5 to 7 tons, giving from seventy to seventy-six blows per minute, with a lift of from 16 to 18 inches. In order to avoid injury to the machine, the hammer is never allowed to fall directly upon its anvil, and, with this object, when not in use, a stop, or gag, is placed between them. When this has to be removed, a piece of iron is placed on the tongue, of sufficient thickness that, on the cam coming in contact with it, the hammer is lifted clear of the prop, which may then be removed and the machine again brought into working order. The foundation usually consists of a solid bedding of wood, containing from 1,000 to 1,500 cubic feet of oak, capped by a cast-iron bed-plate weighing from 10 to 12 tons, and measuring about 24 feet by 7 feet. Two standards, *a*, fig. 93, for carrying the helve, are fixed on the bed-plate in strong jaws, and a third, *b*, for carrying the cam-shaft, *c*, is secured in the same way. The helve, which is T-shaped in plan, is about 8 feet in length, 6 feet in width at the centre of vibration, *d*, 2 feet in depth, and 12 inches wide in the middle; at the end farthest from the point of suspension is a recess 18 inches square, on the lower side, for receiving the hammer-face, *e*. The anvil-block stands on the bed-plate under the centre of the hammer-face, and has, on its upper side, a face, *f*, similar to that of the hammer. The helve is lifted by a revolving cam-ring, *g*, about five feet in diameter, having four cams or wipers on its circumference, which, coming in contact with it, raise it, and passing onwards, allow it to fall upon the bloom resting on the anvil beneath. The following are the approximate weights of the castings employed in the construction of a hammer of moderate size: Bed-plate, 11 tons; helve-stand, 3 tons; helve, $5\frac{1}{2}$ tons; hammer-face, 15 cwts.; anvil-block, $5\frac{1}{2}$ tons; anvil-face, 16 cwts.; standards under cam-ring shaft, $2\frac{1}{2}$ tons; cam-ring shaft, 7 tons; cam-ring, $4\frac{1}{4}$ tons; four cams, 24 cwts.; total, $41\frac{1}{2}$ tons.

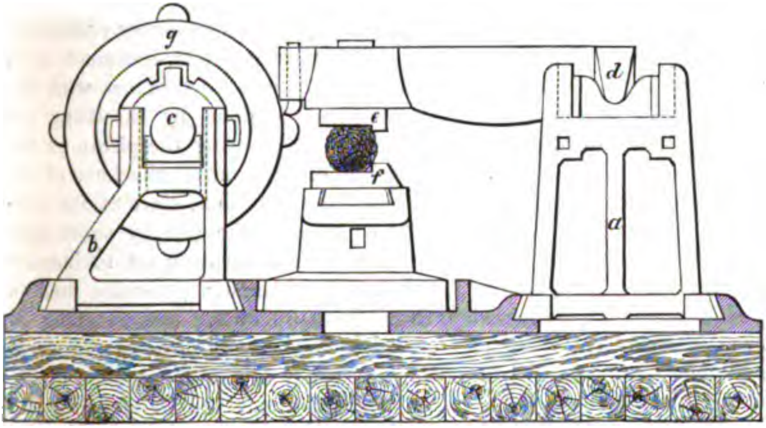

**Fig. 93. – Helve-Hammer, Dowlais; from Truran.**

The puddled ball having been placed on the anvil, the helve is lifted off the prop by a boy, who holds a small iron block beneath the tongue, which, coming in contact with the wiper, the prop is withdrawn and the hammer descends upon the ball. The helve is lifted by the several wipers as they pass in succession, and the ball is converted into a bloom in from eighteen to thirty seconds, during which time it receives from fifteen to twenty blows.

The working-faces of both hammer and anvil are subject to great wear and tear, and require to be frequently replaced; they may be rendered more durable by causing a current of water to circulate constantly through them; but this expedient, introduced by Condie, the inventor of the water-tuyer, was never generally adopted.

The steam-hammer is now generally preferred to the helve for shingling and balling purposes, and is thus employed in nearly all the more recently erected forges. It essentially consists of a vertical high-pressure engine, with an inverted cylinder, supported by a framing, often consisting of two heavy cast-iron standards. The piston-rod, which passes through the lower cover of the cylinder, is directly attached to a heavy block or tup moving vertically between guides on the inner faces of the standards. In single-acting hammers, steam is admitted on one side only of the piston, so as merely to raise the tup, which, on the connection with the boiler being cut off, falls with its whole weight, the steam escaping by an exhaust-port which is opened when the steam passage is closed; double-acting hammers are also made, in which the force of the blow is increased by admitting steam on the upper surface of the piston, and thus accelerating its descent. The steam-hammer possesses a great advantage over those of the ordinary construction, inasmuch as it admits of the force of the blow being regulated in accordance with the requirements of the work in hand. This is done by throttling the exhaust by a properly constructed valve, and allowing the piston to fall upon a cushion of steam. This power of moderating the force of the blow is of great advantage in the shingling of blooms, since at the commencement it is often desirable to consolidate the ball by short light strokes, and afterwards, as the iron becomes more compact, to increase the impact by lengthening the fall. The hammers generally used in puddling forges vary in weight from 30 to 60 cwts.; one weighing 50 cwts. is considered of sufficient power to do the work of twelve furnaces, and may be supplied with steam by a boiler utilizing a portion of the waste heat of the establishment. In steel-works, and for blooming and forging large masses of metal, very heavy hammers, having blocks weighing 80 and even 100 tons, are sometimes employed; those of the largest size are usually single-acting, the employment of steam above the piston being chiefly confined to those of moderate dimensions. Messrs. Thwaites & Carbutt, of Bradford, were the makers of a double-acting steam-hammer, of which fig. 94 is a side elevation. The framing is principally of wrought-iron. The hammer-block, A, which weighs 10 tons, is attached to the piston-rod; the piston is 34 inches in diameter, and the stroke 7 feet. The slide-valve is tubular, and is so balanced against the pressure of the steam as to be easily moved by the lever, *a;* the stop-valve for regulating the admission of steam is worked by another lever connected with the rod, *b.* When not hand-worked, the stroke of the steam-hammer is determined by a tappet coming in contact with the end of a lever which so moves the slide-valve as to allow the steam to escape from below the piston through the exhaust-pipe. The hammer-man standing on the platform, *c*, has the lever, *a*, and another in connection with *b*, close at hand, and at the same time commands an uninterrupted view of the work in process of forging. Small hammers, such as those used instead of the old tilt-hammer in steel-works and for smithy purposes, are frequently made with but one standard, so as to allow of free working on three sides of the anvil, and, in some modern hammers, guides below the cylinder are dispensed with. In that case, the piston is prevented from turning either by an angular piston-rod, or by the use of one of which a portion has been so planed off as to form a flattened surface; this, passing through a stuffing-box having a similar section, prevents any disposition of the piston to turn upon its axis.

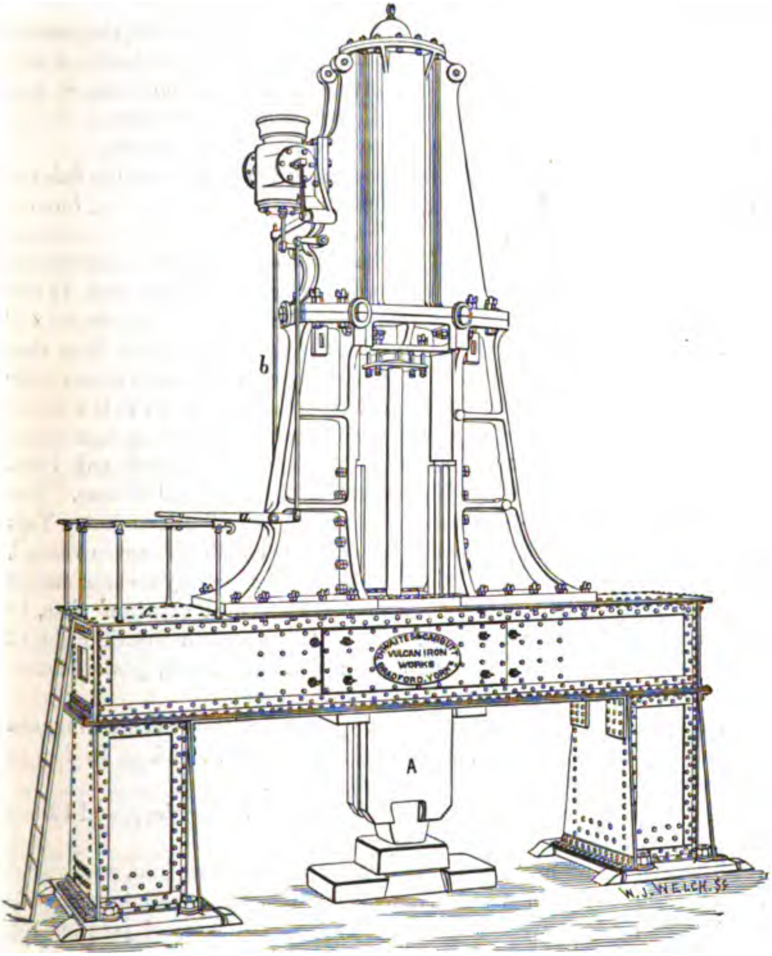

Fig. 94.—Steam-Hammer.

In Condie's hammer the cylinder is cast to the hammer-block, and the piston-rod is suspended to a suitable support by a ball-and-socket joint; large steam-hammers require anvils of great weight, and these should be so arranged as to be entirely clear of the foundations supporting the framing.

A convenient foundation for hammers of moderate size may be constructed of square balks of timber placed on end, and bedded either on concrete or on a mass of cinders broken small, deposited in layers, and well beaten. For the very large hammers used in steel-works, such as Krupp's 50-ton hammer (the maximum lift of which is 10 feet, and of which the anvil weighs 185 tons), the substructure is built of solid blocks of cast-iron; the foundation of this hammer is composed of eight masses of cast-iron, weighing in the aggregate over 1,000 tons.

A 30-ton steam-hammer is in use at the Royal Gun Factories at Woolwich Arsenal. The building containing it and the heating furnaces measures 150 × 100 feet.

The special arrangements of the bed-plates are of a very massive and solid character, and consist in the first place of a hundred 12-inch piles, arranged, at equal distances apart, in the form of a square, 30 × 30 feet. Around and between the piles, for a depth of 4 feet from their heads, is a bed of concrete. Upon the piles is laid a cast-iron plate weighing 115 tons. This plate is in three parts, and upon it is a double layer of 1½-inch elm planks, the upper layer being placed at right angles to the lower one; on these are laid two layers of 12-inch oak balks. Upon these comes a second plate of cast-iron, weighing 150 tons. This plate is cast in two pieces, and covers an area of 27 × 13·5 feet. Then comes a 2-feet thickness of oak timber, placed with the grain vertical, or end on, the collection of balks being held together by an iron strap 6 inches deep by 2 inches thick. These carry a third cast-iron plate, 12 inches thick, weighing 130 tons. Upon this comes a fourth plate, 12 inches thick, weighing 100 tons, a thin packing of oak, just sufficient to prevent contact, being interposed between them.

On the top of the last plate is placed another thin oak packing, and then the round anvil-block, which weighs 103 tons, and is 15 feet in diameter at base, tapering to 12 feet at the top.

Upon this comes a cylindrical anvil, 2 feet 8 inches deep, and 12 feet in diameter, which weighs between 60 and 70 tons.

These foundations include nearly 670 tons of cast-iron, so disposed as to present the utmost solidity, while at the same time retaining sufficient elasticity to prevent any detrimental consequences of jar from the blows of the hammer.

Larger hammers than those described above, namely, of 80 and 100 tons, have been erected at Creusot and Saint-Chamond in France, and lately at Terni in Italy; but even these are found to be unequal to the work of forging very large masses of steel (for which they are now mainly required), and the more efficacious method of compression by hydraulic pressure has now been generally established in large works.

*Squeezers.*—These machines, by which the compression of a ball is effected without impact, are now more frequently employed for the operation of blooming than the old helve-hammer. Squeezers are of two kinds: *reciprocating* and *rotary*. Those of the first class are again distinguished as *single* and *double;* the single squeezer, fig. 95, has but one anvil and one hammer, while in the double squeezer there is a jaw on each side of the articulation, and it has, consequently, two anvils and two hammers.

The lever, *a*, carries a plate of cast-iron, which may be either flat or serrated with parallel corrugations, working against a corresponding fixed jaw, *c*, constituting the anvil. Motion is communicated to this arrangement by the rod, *b*, connected with a crank which is usually attached to the driving-shaft of a rolling-mill. The shingler introduces the ball between the jaws of the machine at their widest part, and gradually moves it forward on the anvil until it comes in contact with the upper jaw, or hammer. At each stroke of the squeezer-arm the ball is flattened by the pressure, and a portion of the cinder is expelled; during the up stroke it is turned over towards the fulcrum of the arm, where it is finally reduced to a bloom of about 5 inches in diameter and 18 inches in length. The up-setting of this bloom, which receives from 20 to 25 successive squeezes during its elaboration, is effected at the extreme end of the jaws, where the distance between them admits of the mass of iron being set on end for the purpose of being pressed longitudinally. The squeezer-crank usually makes about 60 revolutions per minute, and the time necessary for shingling a ball is therefore from 20 to 25 seconds.

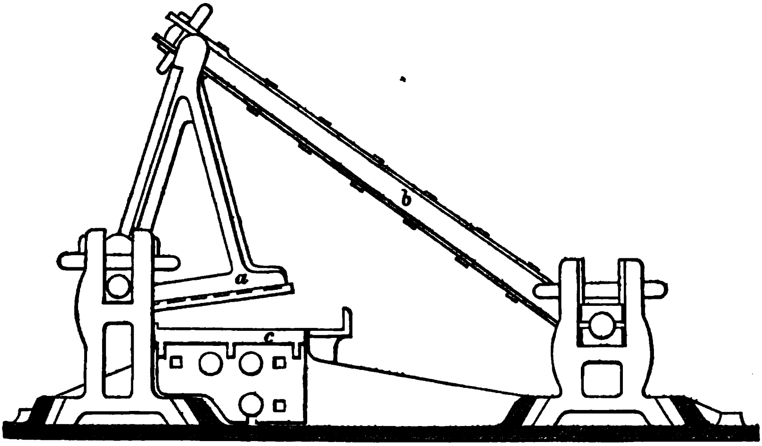

Fig. 95.—Single Squeezer; from Truran.

The rotary squeezer generally consists of a strong cylindrical casting, the inner surface of which is studded with blunt triangular teeth or corrugations; within this revolves a cast-iron cylinder having the outer surface similarly roughened.

The fixed circular casing of cast-iron forms about three-fourths of an entire cylinder, and within this the revolving drum is placed eccentrically with regard to its axis, in such a way that, their surfaces being parallel, the distance between them gradually diminishes in the direction of the line of rotation. The puddled ball enters at the widest part, and on being carried forward by the movable cylinder, is gradually reduced in size until it is ejected at the narrower end in the form of a cylindrical bloom ready for the rolling-mill. The speed of this squeezer is about 12 revolutions per minute, and one machine is stated to be capable of doing the shingling for fifty puddling-furnaces: as, however, there is no means of regulating the distance between the surfaces, it has the disadvantage of requiring that the balls should be as nearly as possible uniform in weight and shape.

*Hydraulic Forging-Presses.*—In the forging of steel ingots, and for other purposes where very powerful compression is required, hydraulic power may be advantageously substituted for hammering. The forging-press of Mr. Haswell, of Vienna, is a machine of this class, and consists of a large vertical hydraulic press with its ram acting downwards against a table serving as an anvil. The large ram is lifted by a small hydraulic press, with which it is connected by a cross-head and side-rods. As the ram rises, the water expelled from the larger cylinder is returned to an accumulator containing a piston, upon the surface of which steam can be admitted; this is employed for moving the ram rapidly when the resistance is not considerable, and the whole power of the machine is consequently not required. When, on the contrary, it is desired to develop the whole force of the apparatus, the connection between the press and accumulator is cut off by a valve, and it is put into communication with a pair of ordinary hydraulic forcing-pumps, driven by a large direct-acting steam-engine.

Fig. 96 represents a large hydraulic forging-press for the heaviest class of steel forgings, constructed by Messrs. Davey Brothers, for Cammell's steel-works at Sheffield.[1] It has two 36-inch pressing rams, A, and two 9-inch lifting rams, B, having a common stroke of 7 feet, whose cylinders are carried by four columns on a heavy entablature, C, of cast-steel. The four rams are connected with the tool-holder or cross-head, D, carrying the forging-tool corresponding to the tup on a steam-hammer, which is guided in its course by slide-blocks bored to fit the columns, E; but the connections are not rigid, spherical bearings being substituted in order to allow for the expansion of the cross-head by heat when the press is at work. The hydraulic power is supplied by three 6-inch single-acting rams driven by two 34-inch steam-engines. Their capacity is such that the cross-head is depressed half an inch for each revolution of the engines; the lifter is raised 8 inches at each revolution, the area of the pressing- and lifting-rams being as 16 to 1. The lifting and lowering of the tool, when not in contact with the forging on the anvil, is performed by water at 60 lbs. pressure per square inch, supplied by the pipe F; the forging-pressure is obtained by the pumps which supply

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[1] 'Engineering,' vol. xli. p. 393.

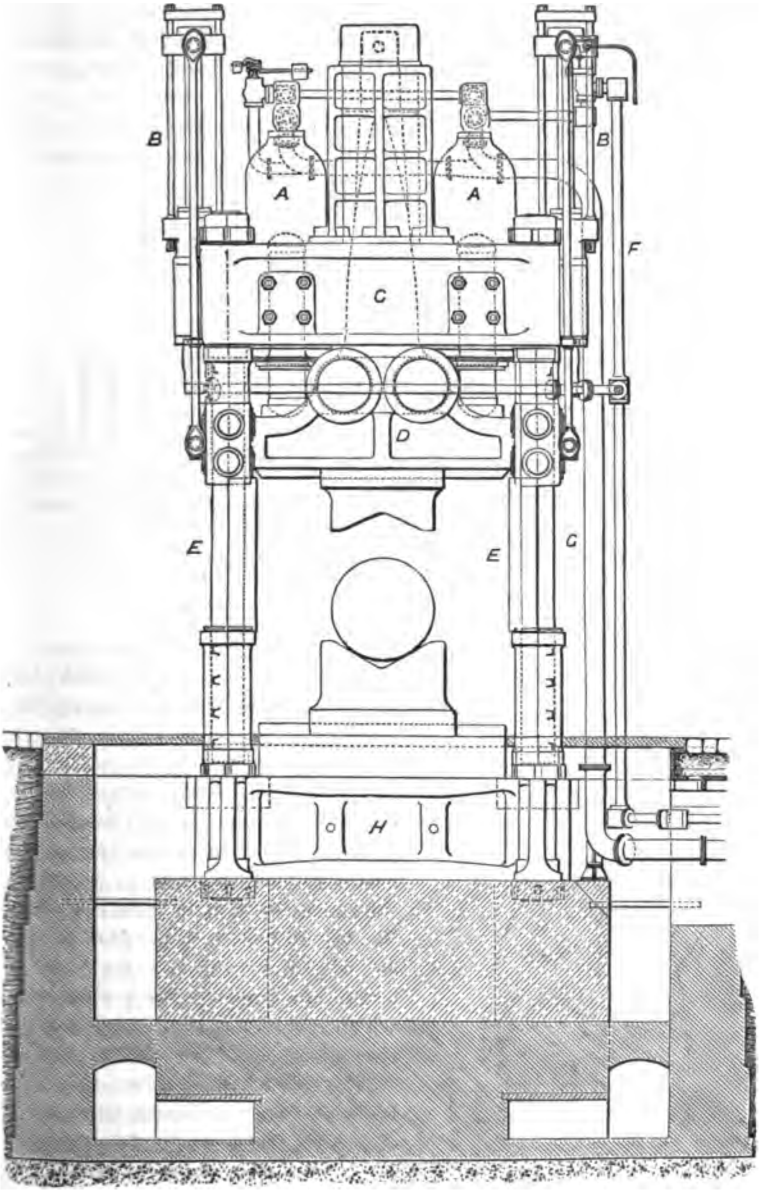

Fig. 96.—Davey's 4,000-ton Hydraulic Forging-Press; front elevation.

the large rams with water at any pressure up to 4,800 lbs. per square inch by the pipe G, the maximum effort being computed at 4,000 tons.

The pumps may be worked up to sixty strokes per minute, and the press is nearly as quick in action as a hammer. The anvil and columns are carried upon cast-steel girders, H, somewhat similar to the entablature, but heavier, which rest upon masonry foundations of considerable thickness, although very much less than would be required for a steam-hammer.

The squeezer, figs. 97, 98, of which the first is a longitudinal section, and the second an end view, was employed with the Danks rotary furnace, and is known as Winslow's, but was improved and modified by Mr. Danks, so as to adapt it to the treatment of very heavy masses of iron. It consists of two corrugated rollers, *a*, each about 4 feet in length and 18 inches in diameter, placed horizontally, occupying one plane, and having the journals fixed in strong frames, *b*. These rollers are made to revolve in the same direction at the rate of from 15 to 20 revolutions per minute, and above them is geared a large eccentric or cam, *c*, the periphery of which revolves at the same rate as the circumference of the two rollers, *a*. At the side of the squeezer-frame is a horizontal steam-hammer, the ram, *d*, of which, seen in fig. 98, hammers the end of the bloom as it is being rotated.

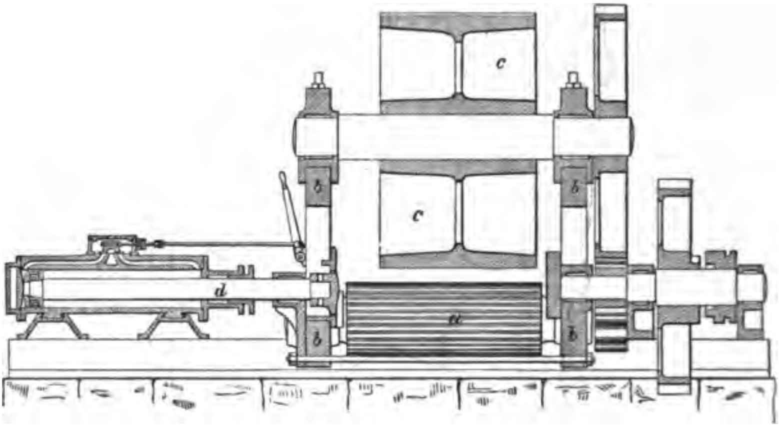

Fig. 97.—Winslow's Squeezer; longitudinal section.

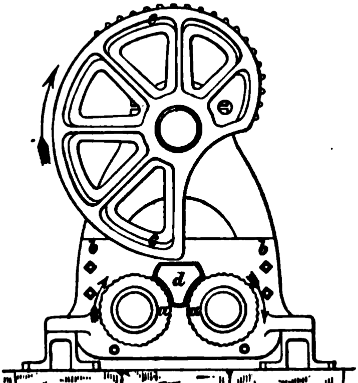

Fig. 98.—Winslow's Squeezer; end elevation.

When the bloom has been sufficiently squeezed, which is effected by two revolutions of the cam, it is removed by a neat lever-arrangement, and rolled upon the floor. It is now seized by a pair of tongs, lifted by a crane used for charging and drawing the re-heating furnace, and placed on a fork, by means of which it is charged. It is subsequently withdrawn by the same fork, placed on a bogie, and taken to the rolls.

The Commissioners appointed to examine the Danks process did not, however, consider this squeezer an essential feature in machine-made iron; and were further of opinion, that if means could be devised for handling under the steam-hammer such heavy masses as balls weighing from 600 to 1,000 lbs., and of getting them worked in a reasonable time, the result would probably be an improvement in the quality of the iron produced.

*Puddling-Rolls.*—Although the hammer is still employed in some of the old open-fire forges of Sweden and Germany for the production of finished iron, it has nevertheless been generally superseded in all the chief iron-producing districts, both of Europe and America, by the rolling-mill. The rolls by which the heated metal is drawn into bars are of two kinds. The first, which are called *puddling-rolls*, serve to consolidate the blooms after their removal from the hammer or squeezer. The second kind, known as *mill-rolls*, are employed for the purpose of extending into bars the masses of iron obtained by cutting puddled bars into lengths, and subjecting them to a welding heat in a balling or re-heating furnace.

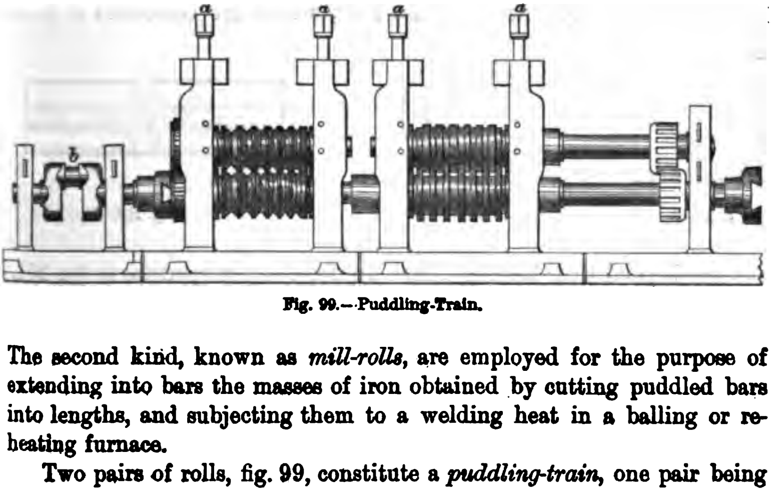

Fig. 99.—Puddling-Train.

Two pairs of rolls, fig. 99, constitute a *puddling-train*, one pair being used for roughing-down the bloom, and the other for finishing it into a bar. The grooves in the roughing pair are either oval, gothic, or diamond-shaped; generally the first two or three grooves are gothic, and the others diamond-shaped. Finishing-rolls are usually turned with grooves capable of producing flat bars from 3 to 7 inches wide and from ¾ to 1½ inch thick. Rolls are supported in pairs one above another in a heavy framework or *housing* of cast-iron, and are so connected by strong spur-gearing as to turn in contrary directions. Motion is communicated to the lower shaft either by steam or by water power, and the distance between the two rolls is regulated by screws, *a*, acting on the brass steps in which their journals or necks are secured. Puddling-rolls are generally from 3 feet 6 inches to 5 feet in length, and from 18 to 22 inches in diameter; the durability of the necks and steps is much increased by the use of cinder-plates, for the reception of which a narrow groove is turned in each roll close to the ends, and a piece of sheet-iron of the proper form is inserted before lowering the top roll. The roughing-down rolls, on the left, have a series of grooves turned on them, which gradually diminish towards the right, and are roughened by indentations cut with a chisel; the finishing-rolls, on the right, have a series of grooves which diminish in the same direction. The lower roughing-roll is provided with a serrated fore-plate and rest, and the bottom finishing-roll with a rest and guide; the crank, *b*, works the squeezer. As the rolls, when at work, are subject to great and sudden variations of torsional strain, the couplings uniting the different members of the train are so made as to have less resisting power than the necks of the rolls themselves; and they are, at the same time, arranged in such a way as to be capable of a certain amount of independent motion. The contrivance by which this is accomplished is shown, figs. 100, 101, and 102, the first being a side view, and the second an end one, of the coupling-box employed. The necks of the rolls, which are continued a short distance beyond their bearings, have the form of the aperture in fig. 101, and slip readily into the coupling-box; one of these is placed on the end of each of the rolls to be joined, the two being united by a loose bar, fig. 102, of similar form, but of somewhat less sectional area, called the breaking-piece or spindle. The collars or coupling-boxes are prevented from slipping by four wooden stops laid in the depressions of the spindle and secured by leathern straps or wire bands. The intermediate shaft, being the weakest part of the train, gives way in case of any undue resistance occurring, and thus prevents the breaking of the rolls. A continuous supply of water is necessary in order to keep the rolls and their bearings cool, and is conveyed through pipes and channels to the various parts where it is required.

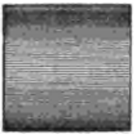

Fig. 100.

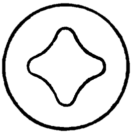

Fig. 101.

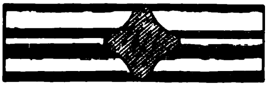

Fig. 102.

On leaving the hammer or squeezer, the bloom, while still at a high temperature, is first passed through the largest groove of the roughing-down rolls, and afterwards, in succession, through the other grooves of both pairs of rolls, until it is finally extended into a long flat bar, of which the surfaces are usually very rough; this is known as "puddled bar," or "No. 1 iron."

Every time the iron has been passed through the rolls it has to be put back again over the upper roll, which is attended with a considerable expenditure of time and labour. Reversing-rolls are sometimes employed to avoid this, so that immediately the iron has passed through, the motion is reversed, and it is passed back through the next groove. Various other contrivances have been resorted to for the purpose of rolling without unnecessary loss of time, and, among them, the most approved appears to consist in the use of two or more pairs set in advance of each other, or in passing the bloom alternately through the grooves of two mills moving in opposite directions; in some cases the bar is carried on an iron carriage, by which it is rapidly taken by steam power from one pair of rolls to the other. Sometimes, and particularly for merchant- and guide-trains, three rolls are placed one above the other, in the same housing. Such an arrangement of the rolls constitutes a *three-high train*, and is driven from the middle; the central roll gearing forward with the lower, and backward with the upper one, or *vice versâ*, so that the bar, instead of being rolled in only one direction, passes backwards and forwards, alternately, between the grooves of the middle and upper, and middle and lower rolls.

The speed of puddling-rolls ranges from 35 to about 80 revolutions per minute; in the Welsh forges the rolls are driven at from 50 to 80 revolutions, but in Staffordshire and Derbyshire they are worked more slowly.

*Shears.*—The puddled bar, on leaving the rolls, is taken by boys to the cutting-shears, which should be placed opposite the finishing-rolls. It is the general practice to shear puddled bars hot; but when the lengths into which they may require to be divided for the *mill-piles* is not known, they are laid aside, to be subsequently cut cold; stronger shears are then required.

The shears usually employed for this purpose consist of two jaws, terminated by cutting-edges of hardened steel, firmly bolted to the iron limbs to which they are attached. The lower blade is immovably fixed to a cast-iron support, while the upper one moves on a pin passing both through it and the cast-iron support of the lower jaw. To the upper limb is attached a lever, which being connected by a strong rod to a crank on a revolving shaft, causes the jaws of the shears to alternately open and shut at each revolution. In this way sufficient power is transmitted to the shears to enable them to divide bars of iron presenting considerable sectional area. Instead of driving shears by means of a crank, or shafting connected with other machinery, when of large size they are generally worked by independent engines. This is the case in the machine represented, fig. 103, where the heavy cast-iron lever, *a*, on which is secured the upper cutting face, *b*, is connected by a sweep-rod with the crank, *c*, on the fly-wheel shaft, and receives motion from the small inverted steam-engine, *d*. Shears are employed for cutting puddled and other bars into lengths for piling, and also for removing the rough ends of finished bars and the edges of plates and sheets. When the length of the cut to be made is considerable, a knife with a diagonal edge, moving vertically between parallel guides, is usually employed. These *guillotine-shears* are much used for cutting boiler-plate, and, as they require considerable power, are driven by engines attached to their framing. For the heaviest class of work, such as dividing steel bars and ingots, guillotine shears driven by hydraulic power are employed.

Rails and other heavy bars have their rough ends removed while still hot, by circular saws; these are from 3½ to 4½ feet in diameter, and make from 800 to 1,200 revolutions per minute. They are generally driven by belts, but in some instances direct-acting steam turbines on the same shaft have been employed.

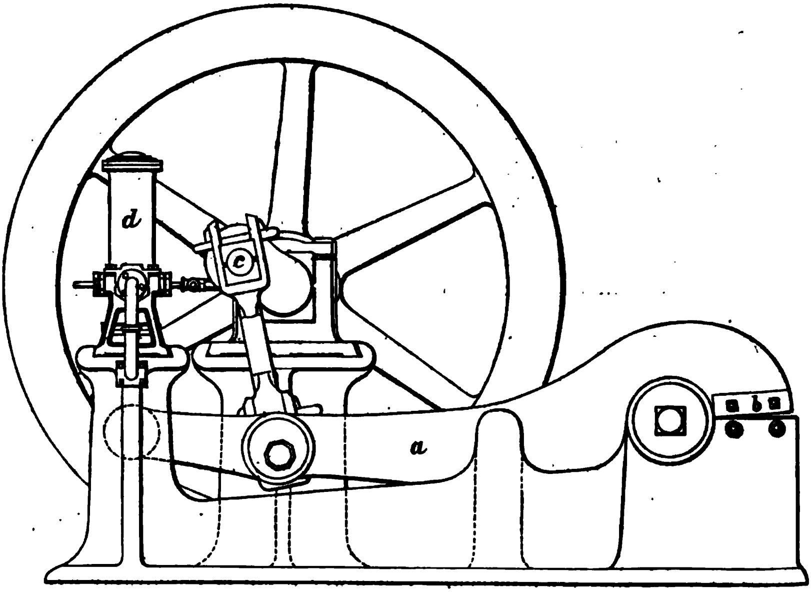

Fig. 103.—Steam-Shears; elevation.

The yield of puddled iron varies considerably in different localities, and will depend not only on the nature of the pig-iron operated on, but to a certain extent also on the skill employed in its treatment. In the neighbourhood of Dudley, South Staffordshire, the ordinary calculation is, that 24 cwts. of pig-iron should yield 22 cwts. of puddled bars, and that about 1 ton, 2 cwts., 2 qrs. puddled bars are employed in the production of 1 ton of merchant-bar. In South Wales, in 1859, it was estimated that 27 cwts. of white pig-iron were required to produce 1 ton of finished or merchant-bar.

WORKING PUDDLED BAR INTO MERCHANT IRON—THE MILL.—After having been cut by shears into suitable lengths, the puddled bars are piled in packets, which are heated to a welding heat, and then hammered and afterwards rolled, or they are at once rolled into bars without hammering. The elevation to a welding temperature is effected in special furnaces, known as mill-, balling-, or re-heating furnaces.

*Re-heating or Balling.*—The re-heating furnace very closely resembles the puddling-furnace, and has a chimney, *a*, of similar dimensions, but is usually 8 or 9 inches wider, and about two feet longer; the average area of the fire-place, *b*, is 12 square feet.

The dimensions, form, number, and size of doors, &c., of the re-heating furnace vary considerably, in accordance with the nature of the work for which it is to be employed; but the following woodcuts, figs. 104 and 105, after Truran, represent a longitudinal and a horizontal section above the level of the hearth of a furnace such as is commonly employed in South Wales for the conversion of puddled bars into rails or merchant iron.

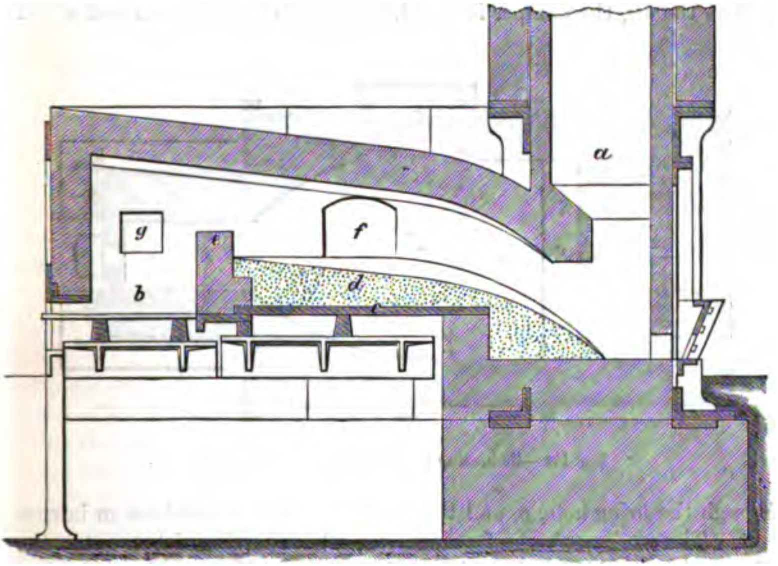

Fig. 104.—Re-heating Furnace; longitudinal section.

The cast-iron bottom, *c*, is about 14 inches below the working-door, and on it is laid a sand bottom, *d*, falling from the door both towards the back of the hearth and towards the chimney. Many re-heating furnaces are constructed without an iron bottom, and in such cases the material forming the hearth is laid on rubble-work, consisting of old fire-brick, fire-brick ends, &c. Between the hearth or body of the furnace, and the fire-place, a bridge, *e*, 9 inches in thickness, is carried up to within 14 inches of the roof, while at the stack end the sand is gradually rounded off so as to meet the bottom of the flue.

A number of puddled bars, generally from 3 to 4½ feet in length, are placed together to form each *pile*, of which the sectional area is from 3 to 10 inches square, in accordance with the size of the iron to be made; piles 3 feet 6 inches long, 7 inches wide, and 8 inches high, are a common size for the larger descriptions of merchant iron. The baller charges four of these for a *heat* through the door, *f*, by placing them singly on a flat iron bar, called a *peeler*, and slides them into the furnace, taking care not to disturb the arrangement of the bars. When charged, the four piles lie nearly across the furnace, radiating from the door; the ends towards the back lying about 6 inches lower than those nearest the door.

The door, *f*, is now closed, and a little fine coal thrown around it to exclude the air, the damper is raised, the grate cleaned, fresh fuel added through the firing-hole, *g*, and the fire urged, so as to produce an intense heat. The workman's chief occupation, after charging, is to watch the piles, and to so shift their positions that they may be equally exposed to the fire, and be brought to a welding heat in the shortest possible time. As this point is approached, the iron becomes externally oxidized, and forms a scale which covers the surface of the pile, and which, by combining with the siliceous matter of which the bed is composed, forms a slag, which, running off freely towards the bottom of the stack, escapes from the furnace. This is distinguished by the name of *flue-cinder* from that produced in the puddling-furnace, which is known as *tap-cinder*. A small fire is usually placed in front of the stack of re-heating furnaces to prevent the tap-hole from becoming obstructed by the cooling of the cinder. At the expiration of sixty minutes a heat such as that described will be ready, and the piles are then successively grasped by a pair of heavy tongs, and placed on a bogie, to be carried to the rolls. The withdrawal of the piles, charging a fresh heat, and repairing the bottom, will

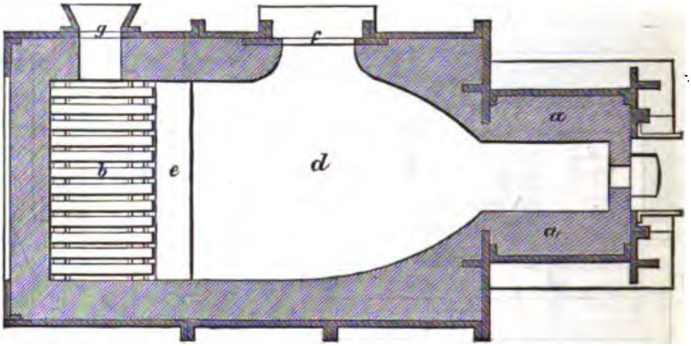

Fig. 105.—Re-heating Furnace; horizontal section.

occupy about sixteen minutes; such piles usually average about 4 cwts. each, and consequently a furnace working thirty-six piles in the course of twelve hours will get through 83 tons of iron weekly. For smaller descriptions of merchant-bar, the piles are made about 18 inches long, 3 inches wide, and 2½ to 2 inches deep. Sixteen or eighteen such piles, which take from twenty-eight to thirty minutes to reach a welding heat, are charged at once; the time occupied in withdrawing the heat, repairing and re-charging, is about twenty-one minutes, and a furnace working on piles of this description will re-heat 31 tons of iron per week. Bars of the smallest size are rolled from bolts of manufactured iron called *billets*, measuring from 12 to 20 inches in length, and having a diameter of from 1¼ to 1¾ inch; for these a smaller furnace is employed, and from twenty-five to thirty billets are heated at a time. Cold billets are introduced as fast as hot ones are withdrawn; furnaces working on billets for guide-iron will heat from 15 to 25 tons per week, according to the size of the finished bars.

The ordinary weight of the piles for rails is, in South Wales, about 15 cwts.; four of these are placed in the furnace at once and the whole heat is rolled into blooms in a triple mill in five minutes. After a second heating, which occupies about thirty minutes, the blooms are each passed nine times through the rail-mill, and become rails. The loss on the piles, including crop-ends, which are subsequently utilized, is about 20 per cent.

The amount of labour bestowed on the manufacture of merchant iron varies with the quality it is intended to produce. For the commoner descriptions it is usual to pile puddled bars, or No. 1 iron, cut into proper lengths, and these, when brought to a welding heat, are rolled into bars, either with or without being previously worked into blooms under the hammer. More frequently, however, No. 2 iron, or that which has been twice rolled, is used for the top and bottom bars of the pile, when best iron, or No. 3, is being made. If, after this, the iron be further piled and welded, it is distinguished as *best-best*, and *treble-best;* according to the number of re-heatings and rollings to which it may have been subjected.

As a rough approximation, it may be estimated that the amount of coal consumed for the manufacture, from the ore, of common finished bars, of No. 2 quality, is about four times their weight: to this must be added from 9 to 10 cwts. per ton for each additional heat to which the iron may have been subjected.

The bottoms and tops of rail-piles are sometimes covered by slabs made by doubling and welding together, under the hammer, two or more puddled blooms, which are then re-heated and rolled, without first having passed through the intermediate state of puddled bars. The necessity for the use of single slabs for the outside of piles arises from the circumstance that butt joints, unless covered, do not weld properly: it is also necessary that the ends of the bars forming the pile should be cut square, and that all contact-surfaces should be as free as possible from scale and rust.

Gas re-heating furnaces are in modern iron-works practice very largely used in the production of forgings and finished iron. Of the different styles employed, that on the regenerative principle by the late Sir W. Siemens has been described and illustrated, pp. 104–5, figs. 25, 26, and 27. This furnace has been very widely applied, but in many cases the system of conducting the producer gas directly to the furnace, so that it may be delivered hot, and the air only requires heating, is now adopted in preference to the use of a cooling-tube with subsequent heating in regenerators. The air for burning the gases may be heated in pipes, but a more perfect system is to use a shallow flue of the whole breadth and length of the furnace, which cools the bottom at the same time that the air is warmed. This is adopted in Bicheroux's furnace, which is now more particularly used for heating steel ingots. In Ponsard's continuous regenerative system, the air is heated in a chamber below the furnace filled with brickwork, arranged in such a fashion that there are two independent systems of passages, isolated from each other. Through one of these the waste flame passes in its way to the chimney, while through the other the air travels in the opposite direction. This, though not quite so efficient as the Siemens system, has the advantage of being simple both in construction and working. The Ponsard furnace is continuous instead of intermittent; and as the heating medium is spent gas only, no explosion can take place in the event of air getting to the flame-flue through leakage in the brickwork.

*Mill-Rolls, &c.*—The quality of bar-iron is much improved by hammering, since the rapid consolidation which takes place under the heavy blows of a steam-hammer expels the cinder, while the iron is at a sufficiently high temperature to allow of its escape. A large portion of the cinder is, however, eliminated during the operation of rolling the pile into a bar; but in consequence of the great reduction of temperature which has taken place before the last groove has been reached, a certain amount of slag is liable to become enclosed in the iron. Hammered iron is more homogeneous, has a greater specific gravity, and is superior in point of strength to that which has not been thus treated; consequently in the manufacture of the best qualities hammering should not be dispensed with.

A train of mill-rolls for large iron consists of two pairs; one for roughing, which may be about 6 feet 6 inches long by 22 inches in diameter, and the other for finishing, considerably shorter and of somewhat less diameter. The whole of the plant requires to be as strong and substantial as for the puddling-train, but the standards of the finishing-rolls are provided with various tightening and adjusting screws for maintaining them accurately in their positions. Motion is communicated to the finishing-rolls by a pair of pinions and spindles, while from the bottom roll a coupling-spindle communicates motion to the bottom roughing-roll, whence it is transmitted to the upper one by spur-gearing, keyed on the ends of the pair. This method of driving from the lower roll possesses the advantage of permitting the use of larger or smaller roughing rolls, as may be required.

In three-high trains the lifting of the piles from the lower to the upper level is easily effected when light bars only are being rolled, but in the case of heavy piles it is attended with considerable labour and loss of time, unless some special mechanical appliance is employed. The usual method adopted is to make the feed-plates or tables movable, and to so connect them with a single-acting steam- or water-pressure engine, that the pile, after passing through grooves between the lower rolls, is lifted to the upper ones, and, after having passed between them, is received on a table on the other side, which at once descends to the level of the lower pair. A similar arrangement is also often used for heavy plate-mills consisting of a single pair, since the pile, after having passed between the rolls, has in this case only to be deposited on top of the upper roll to be again carried back to the side from which it entered.

For rolling bars of small section, which from their flexibility are liable to become twisted, it is usual to use a three-high train, to the tables or aprons of which guide-jaws or friction-rollers are attached; these, which are employed for keeping the ends of the bars straight on entering the grooves, give their name to the arrangement, which is known as a *guide-train*.

Wagner's rolling-mill, of which fig. 106 is a front elevation, is sometimes called "the universal mill" on account of the facility with which it may be made to produce bars and flats of various sizes with the same pair of rolls. This machine consists of two horizontal rolls mounted and geared in the usual way; to these is added a pair of vertical rolls, *a*, working in bearings, which can be traversed horizontally on slides by right and left screws. The simultaneous motion of these screws is insured by the hand-wheel, *b*, geared to a shaft carrying two worm-pinions acting on wheels keyed on the screw-spindle. By turning these the two vertical rolls may either be brought nearer together or removed farther apart; thus regulating at will the width of the bar to be produced. The vertical rolls, *a*, receive their motion from the driving pinion through mitre wheels geared into similar wheels, sliding on their shaft in such a way as to follow horizontally the movements of the vertical rolls. The horizontal top roll is kept to its work by a pair of counterweights, *c*, and its distance from the bottom roll is regulated in the usual way by a pair of screws geared to the hand-wheel, *d*. A combination of this kind, under the name of "White's mill," has been advantageously employed in South Wales for roughing rail-piles.

Very heavy mills, such as those used for rolling armour-plates, are reversed at each passage of the pile. In Ramsbottom's rolling-mill the rolls are driven, without the intervention of a fly-wheel, by a pair of direct-acting horizontal engines, coupled at right angles, which are reversed, after each pass of the bloom or ingot, by hydraulic pull-over gear; the rolling is thus performed alternately in opposite directions. The motion is transmitted from a pinion on the crank-shaft of the engine to a spur-wheel in connection with the rolls, which make one revolution for three and a quarter revolutions of the engine when cogging or breaking down ingots; but in finishing-mills either the gearing is equal or the engine is directly attached to the rolls.

The size and speed of rolling-mills vary within very wide limits, according to the nature of the work to be performed; reversing-mills for heavy plates make only from 25 to 30 revolutions per minute, while some very small mills employed for special purposes make over 500 revolutions in the same time. For ordinary-sized merchant-bars, the diameter of the rolls is from 12 to 18 inches, and the number of revolutions per minute from 80 to 110. Rolls for roughing rail-piles are usually from 20 to 24 inches in diameter, and, if worked as a reversing train, the speed does not exceed 30 revolutions; if not reversing, the number of revolutions varies from 80 to 100 per minute.

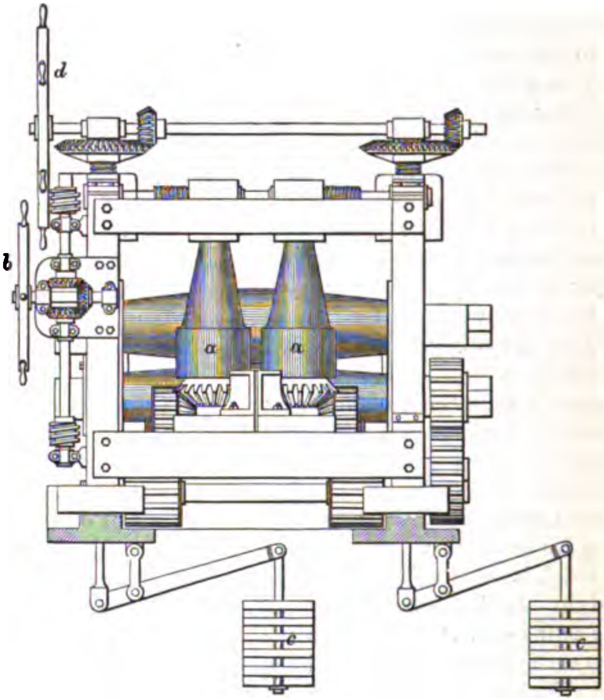

Fig. 106.—Wagner's Rolling-Mill; front elevation.

*Plates and Sheets.*—The rolls employed in the preparation of plates and sheets are of a plain cylindrical form, of the same diameter throughout, and, in order to increase the hardness of their surfaces, they are invariably cast in chills. The distance between the rolls is diminished each time the pile is passed through, and the top roll requires to be supported to prevent its coming in contact with the lower one when running light. This is done by supporting the lower step of the top roll on the ends of a forked rod connected with a lever and weight, so as to slightly overbalance it. For the purpose of securing accuracy of adjustment and insuring the perfect parallelism of the two rolls, toothed-wheels are attached to the heads of the setting screws, which are moved through equal spaces by spur- or bevel-gearing, and carefully divided hand-wheels. For rolling tapered iron, the setting screws are sometimes provided with self-acting gear, by which the distance between the rolls is gradually and uniformly diminished during the passage of the pile.

Armour-plates for ships, made up to 12 inches in thickness, and other very heavy plates, may be produced either by hammering or rolling alone, or by a combination of the two operations. The material employed for the manufacture of hammered plates is best scrap-iron, which is balled, re-heated, and welded, until a slab is obtained somewhat thicker than the section of the finished plate, and of which two of the edges are square and the other two chamfered. These are welded together with their tapered edges, on the shorter sides, overlapping, so as to form a section of a plate of the required breadth, and finally, the length is made up by the addition of so many pairs as may be required. For convenience of handling, a *porter-bar* or *staff* is welded to the unfinished plate, and this being provided with a capstan-head with levers, and supported by a crane, allows of its being readily turned on the anvil as required. The finishing of the plate and its reduction to the proper thickness are effected at a moderate red heat, and water is constantly thrown on its surface to facilitate the removal of scale. When finished, it is heated to redness and annealed by slow cooling.

Rolled armour-plates, instead of being built up edgewise, are formed by the successive superposition of slabs, re-heating, and re-rolling. For the finished plate, large slabs, each about 8 feet long, 4½ feet wide, and 2½ inches thick, are piled and introduced into a furnace, in which they are placed on fire-brick pillars, so as to allow the flame to circulate beneath them. The door of the furnace is on the side parallel to the axis of the rolling-mill, and when the pile has become sufficiently heated, it is transferred to a truck, which takes it directly to the mill. After passing between the rolls, it is received on a similar truck on the other side, and is passed backwards and forwards by reversing the rolls until sufficiently reduced in thickness. The tops of these trucks are either inclined towards the rolls, or they receive the plate upon friction-rollers in such a way that it may be readily pushed forward; the action of the mill being sufficient to force it up the inclined surface on the opposite side. After leaving the rolls, and while still hot, the plate is rendered perfectly flat by the passage over it of a roller weighing about 7 tons; when cold, it is transferred to the table of a planing-machine, where all its edges are dressed square.

Plates and sheets are classified in accordance with their thickness: the former term embraces all strengths exceeding No. 4 of the Birmingham wire-gauge, corresponding to a thickness of 0·238 inch; all less thicknesses are sheets. Sheet-iron is classified as follows:—

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Singles | including from No. | 4 | to No. | 20 | gauge | 0·238 | to | 0·035 | in. thick. |
| Doubles | ,, | 20 | ,, | 25 | ,, | 0·035 | ,, | 0·020 | ,, |
| Trebles, or lattens | ,, | 25 | ,, | 27 | ,, | 0·020 | ,, | 0·016 | ,, |

For the heavier classes of plates, the piles are built up of bars, which, instead of having their longer sides parallel, cross each other alternately; the coverings at top and bottom being flat slabs, from 9 to 14 inches in width and from 1 inch to $1\frac{1}{4}$ inch in thickness, made by doubling two puddled blooms under the hammer and rolling, at a heat, to the proper size.

The pile for plates, which when finished measure 6 feet in length, 3 feet in breadth, and have a thickness of $\frac{3}{16}$ inch, is about 20 inches long, 7 inches high, and 12 inches broad. It is first reduced to a roughly squared bloom by passing it lengthwise through three grooves in the blooming-rolls, then four times in the direction of its breadth through the plate-roughing rolls, and finally, three times lengthwise through the finishing-rolls.

The pile for sheets of large size, such as singles of No. 12 gauge, which are 6 feet in length by 2 in width, weighing about 70 lbs. each, is made up of scrap and crop-ends produced in making top and bottom plates. About twenty such piles are placed in the furnace at one time, and each is first rolled into a bar 3 feet 6 inches in length and 7 inches in breadth, and then cut transversely into two equal parts. Each of these portions is now passed through the roughing-rolls the wide way, until it has assumed the form of a plate having nearly the required thickness, and of a width represented by the length of the original half-pile. After being passed lengthways three or four times through the finishing-rolls, the two sheets, produced from the halves of the original bloom, are passed through together three or four times; they have now become nearly cold, and after being softened by heating to low redness in an annealing-furnace, are cut to the proper size and finished.

Thinner sheets are rolled in a similar way, excepting that they are made from smaller piles; in rolling lattens, after the first annealing heat, four plates are passed through together, and, after the final heat, eight thicknesses are passed at the same time. The thin sheets, or *black-plates*, intended for tinning are made in a similar way, but as the unfinished work is doubled after every heating, as many as sixteen thicknesses are at last passed through at the same time; they are finally cut to their proper sizes, pickled in weak sulphuric acid, and their surfaces polished by cold rolling.

For the production of small square bars, such as *nail-rods*, the slitting-mill, in which the rolls are replaced by arbors carrying steel discs, is commonly employed. In this arrangement the discs on the upper arbor interlock with those on that which is beneath it, thus constituting a rotary shearing-mill with several pairs of cutters. When a flat bar of iron is passed between these cutters, in the same way as in an ordinary rolling-mill, it is divided into slips or rods of rectangular section, which are delivered on the other side in a somewhat bent and twisted condition, from the pressure of the cutters; these are afterwards straightened and made up into bundles for the use of nail-forgers. The bar is steadied, while passing between the cutters, by guides, and a tank above the framing contains water, which is allowed to fall in several small streams upon the slitters, for the purpose of keeping them cool.

Iron made from ores containing a considerable amount of phosphorus is always *cold-short;* thus, the characteristic of Cleveland iron is cold-shortness. Cold-short iron is also produced when siliceous materials are used for fettling; both phosphorus and silicon, therefore, appear to make iron cold-short.

The exact cause of *red-shortness* in welded iron is not always very clear. Red-short iron is produced from ores that are deficient in phosphorus, and the addition of that substance, by the introduction of Cleveland pig, or some similar variety, produces the best results, when mixed with hæmatite pig or the purer charcoal irons. The red-short character of Welsh iron cannot always be attributed to the absence of phosphorus, as it is in many instances, although not always, undoubtedly due to the presence of sulphur. Iron may at the same time be both red-short and cold-short; this, which is the worst possible description of iron, is produced from ores containing a high percentage of both sulphur and phosphorus. Red-shortness is believed to be, in some cases, due to a deficiency of carbon in the wrought-iron, since the most fibrous and the toughest iron acquires this property if melted in a clay crucible and afterwards heated, doubled, and welded.

The waste heat of puddling and re-heating furnaces is frequently employed for the generation of steam, and it is sometimes also made use of for heating the blast; another method of utilising this waste heat has been described under the head of "Gaseous Fuel."

## STEEL.

Every description of ironstone is capable of affording cast-iron and wrought-iron; but the properties of the product obtained will vary both in accordance with the nature of the ores employed, and with the method of treatment resorted to, since all will not effect, in an equal degree, the elimination of foreign substances. The metal obtained will consequently be more or less tenacious or brittle, hard or soft, pure or impure; in all cases, however, the names "cast-iron" and "wrought-iron" will be applied to the extreme results. In the same way all intermediate products which cannot be classed with cast-iron on the one hand, or with wrought-iron on the other, may be called "steel."

The crude product obtained from the reduction of iron ores in the blast-furnace is known as "cast-iron;" it is not malleable, particularly when hot, but may be hardened by sudden cooling. The term "wrought-iron" is applied to the more or less refined metal produced either from pig-iron or directly from iron ores; it is malleable, both hot and cold, but it is not capable of being tempered.

Steel forms an intermediate link between ordinary cast-iron and wrought-iron, uniting, in a certain degree, the properties of both; its distinguishing characteristic is its capability of being hardened or softened at pleasure by rapid or slow cooling. It cannot, however, be said where steel begins or where it ends; it is a member of a series commencing with the most impure pig-iron, and ending with the softest and purest malleable iron.[1]

According to Frémy, steel is not simply a combination of carbon and iron, but is a nitro-carbide, the presence of nitrogen being essential to the production of steel. He, however, subsequently became aware that not only does wrought-iron contain nitrogen, but also that it is present in larger proportion than in ordinary cast-steel; and, as the case at present stands, the weight of evidence is decidedly against the necessity of the presence of nitrogen in steel. The older view of Karsten, that its essential qualities are due to variations in the amount of carbon present, is now generally admitted to be more probably correct. At all events, it is now thoroughly established that nitrogen occurs in steel in very small proportions only; and that if its presence be necessary to the constitution of steel, it must be still more necessary to that of wrought-iron and pig-metal.

Steel may be produced—first, directly from iron ores; secondly, by the addition of carbon to malleable iron; thirdly, by the partial decarburization of pig-iron; fourthly, by diluting the carbon in pig-iron by the addition of malleable-iron.

STEEL BY THE DIRECT REDUCTION OF IRON ORES.—By the Catalan process, previously described, steely iron, *fer aciéreux*, is produced under certain special conditions of working. The most important of these conditions are as follow: The employment of a small quantity of greillade, and a large proportion of charcoal; the ore is frequently and gradually pushed forward from the contrevent towards the tuyer; the slag is tapped often, and ample time allowed for the formation of the "massé" or bloom. It is also generally considered that the tuyer should be less inclined, and that the contrevent should have more slope; towards the end of the process less blast should be given than in the case of soft iron. The denser varieties of charcoal should be employed, and the presence of manganese in the ores treated is desirable. Much will, however, depend on the skill of the workman; as, with the same materials, one man will produce a large proportion of steely iron, while another will obtain little or none.

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[1] Karsten gave this definition of steel as long ago as 1823. "Annales des Mines," 1824, vol. ix. p. 657.

Although in this case steel is produced in one operation, yet the ore must first be reduced, and the resulting metallic iron subsequently carburized by contact with incandescent charcoal. By this process uniform carburization cannot be secured; the bars, obtained by hammering out the resulting blooms, are therefore broken on the anvil, and the various fragments selected and classified in accordance with their respective fractures.

A patent was granted in 1791 to Samuel Lucas for making cast-steel by melting rich iron ores with carbonaceous matter—charcoal, horn, bone-dust, or other cementing substances. Patents were obtained for substantially the same process by David Mushet in 1800, and by John Isaac Hawkins in 1836. In 1854 a patent was granted to Samuel Lucas for an improved method of manufacturing steel, which essentially consisted in interstratifying the bars of iron in an ordinary converting furnace with lumps of iron ore. It is directed that the bars should not be allowed to touch the iron ore, as they would adhere to it; and a claim is inserted in the specification for the conversion of iron ore into steel without the presence of bar-iron.

A patent was obtained by William Edward Newton, in 1856, for improvements in the process of manufacturing steel, &c., being a communication from abroad. The iron ores are directed to be reduced to fragments of about forty to the cubic inch, mixed with charcoal or other carbonaceous matter, and, if necessary, with suitable fluxes, in alternate layers, and kept heated to whiteness during about forty-eight hours in a suitable cementing vessel. After cooling, the ore thus treated is either melted in crucibles, to form cast-steel, or worked up in a furnace into spring-steel.

Some years since experiments were made at the Dowlais Iron-Works by Mr. E. Riley, on the direct production of cast-steel from iron ores; it was, however, found, that although steel of excellent quality was sometimes produced, uniform results could not be obtained.

STEEL BY ADDITION OF CARBON TO MALLEABLE IRON. *Cementation.*—This is an old process, but of its history little is known. Beckmann states that there is no allusion to it in the writings of the ancients; but it was well described in 1722, by Réaumur, in his treatise on the art of converting bar-iron into steel. This treatise is illustrated with engravings, in which converting-furnaces, similar in all essential respects to those now in operation, are represented.

When it is desired to purify cast-iron as completely as possible, the operation of fining must be prolonged until wrought-iron is produced, and from this steel is obtained by re-carburization. This is the method usually pursued in the Bessemer and Siemens-Martin processes for the direct production of cast-steel; but, as in these operations the re-carburization of the iron is effected by the introduction of cast-iron, the impurities contained in it become incorporated in the ultimate product. When, therefore, steel of fine quality is required, the carburization must be effected on the principle of cementation, by the use of carbonaceous reagents.

The furnace in which this operation is conducted, is represented in the accompanying woodcut, fig. 107. It consists of an oblong rectangular chamber, divided into two parts by a long and narrow fire-place, *a*, which passes through its centre and is provided with a door at either extremity, by which fuel is supplied. On each side of this is a chest or *converting-pot*, *b*, made either of fire-brick or fire-stone, and so supported on flues as to allow of the heat and flame passing beneath the troughs through openings, in connection with the chimneys, *c*. By these the smoke and heated air escape from beneath the arch, which is thrown over the two chests and the fire-place by which they are heated. In the brickwork at the ends of these troughs man-holes, *d*, are left for the purpose of introducing the iron bars into the furnace; these are bricked up during the working of the apparatus, but when it has cooled they allow of the workmen entering either to charge the bars of iron, or to remove the steel produced by their cementation.

The whole furnace is built under a conical hood, *e*, of from 30 to 40 feet in height, which serves both to prevent loss of heat by radiation, and to carry off the smoke and gases generated by the combustion of the fuel employed.

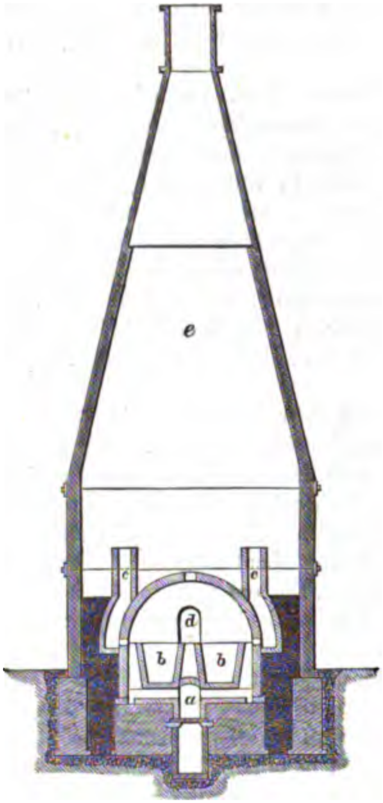

Fig. 107.—Converting-Furnace; transverse section.

The converting-pots vary from 8 to 15 feet in length, and from 2½ to 3 feet in width and depth; the smaller chests are found to produce steel of more uniform quality, but are less economically worked than those of larger size.

The depth of the fire-place depends both on the nature of the fuel employed, and on the dimensions of the chests to be heated: the space between these is usually about 18 inches in width, but in some instances one chest only is employed, and under these circumstances it is placed immediately over the grate on which the fuel is consumed. The degree of heat applied is regulated by opening or closing apertures in the arch, and by limiting the amount of air passing into the furnace through the grate.

The iron to be converted is in the form of straight bars, usually about 3 inches in width and $\frac{3}{4}$ inch in thickness; in order to allow for expansion these must be somewhat shorter than the chest in which they are to be placed. Charcoal, which has passed through a riddle, of $\frac{1}{2}$ to $\frac{3}{4}$ inch mesh, is first spread evenly over the bottom of each chest, and on this a layer of bars is laid longitudinally, flat side downwards, with very small spaces only between them. When the iron is too short to extend the whole length, the empty spaces are filled up with short pieces or ends of bars. This first layer of bars is covered by a stratum of charcoal about half an inch in thickness; on this another layer of bars is placed, and so on in succession, iron bars alternating with layers of charcoal, until the chests are filled. A thick layer of charcoal is now placed upon the top, and the whole is plastered over with grinders' waste, or "wheel-swarf," a substance produced by the wear of the siliceous grindstones employed by cutlers and others in the manufacture of articles of steel. This consists of disintegrated siliceous sandstone mixed with finely divided and partially oxidized particles of steel, which combine more or less completely with the silica, and a compact air-tight covering is the result; when grinders' waste is not obtainable, clay may be substituted for it. As soon as the charging of the chests has been effected, the man-holes are stopped with bricks, and the fire is lighted, care being taken to keep up a temperature of glowing redness during periods varying in accordance with the nature of the steel it is desired to produce. *Spring-steel* requires seven days, *shear-steel* eight days, and steel for welding from nine to ten days. The progress of the operation is, from time to time, ascertained by means of trial-bars, which are inserted and removed through holes left in the ends of the chests, and from an inspection of the fracture of these, when cold, a judgment is formed of the degree of carburization which has been attained. The ends of these bars protrude beyond the furnace, and care is taken to prevent access of air by carefully claying-up the openings left between the iron bars and the sides of the trial-holes. When the cementation has attained the desired point, fuel is no longer supplied to the grate, and the furnace is allowed to cool during several days before commencing to remove the charge. As soon, however, as the temperature has become sufficiently reduced to allow a man to enter, the bars are taken out, broken, and assorted in accordance with the indications presented by their fractured surfaces. Iron produced from Swedish magnetic ores is employed in the production of the best kinds of cement-steel, and, generally speaking, hammered bars are preferred to those made by rolling. The smaller forges, situated in the eastern part of Sweden, and working in connection with the Dannemora mines, produce the most esteemed brands; the iron of Löfsta, known as Ⓛ iron, is one of those having a very high reputation. The charge of a furnace of the usual dimensions consists of from 16 to 18 tons of bar-iron; as before stated, chests of moderate size are found to afford more satisfactory results than very large ones, since in the latter, a uniform temperature cannot be maintained throughout. Consequently, the bars towards the centre will be carburized in a less degree than those situated nearer the bottom, sides, and top of the chest.

On "drawing a heat" a portion of the charcoal is always found to have retained its original form, while the remainder has become reduced to a soot-like dust; the whole is now sifted and washed, in order to free it from the finer particles, and, when dry, the portion which has not passed through the meshes of the sieve is added to an equal bulk of fresh charcoal, and well mixed with it. The mixture thus obtained is found to afford more satisfactory results than entirely fresh charcoal, which requires a longer time to effect the complete conversion of the iron. The addition to charcoal of small quantities of carbonate of barium, alkaline carbonates, ferrocyanide of potassium, or of organic matter containing nitrogen, has at different times been recommended, but none of these substances are practically in use, except for case-hardening.

The physical properties of the bars before and after conversion differ very considerably; the colour of the fractured surfaces of the carburized bar has no longer the bluish tint of malleable iron, but has acquired a reddish-white aspect not unlike that of bismuth, and the texture has become scaly and crystalline.

The most remarkable characteristic of the carburized bars, and that from which this product derives the name of *blister-steel*, is, however, the blistering of their surfaces. When the blisters are small in size, and are distributed with a certain amount of regularity, it is an indication that the steel is of good quality; but when, on the contrary, they are large and follow particular lines, it is indicative of a want of homogeneity in the iron used.

Much diversity of opinion has been entertained with regard to the cause of these blisters, which are evidently due to gaseous expansion from within, while the iron, from being exposed to a high temperature, is in a soft state. The most probable explanation appears to be that they are due to the action of the cementing material on particles of slag, consisting of ferrous silicates, enveloped in the metal, and, as the reduction of the iron in these to the metallic state will be attended with the formation of carbonic oxide, the evolution of this gas would account for the formation of the blisters.

The average increase of weight experienced by iron during its conversion into blistered steel amounts to from $\frac{1}{2}$ to $\frac{3}{4}$ per cent.; and the amount of coal consumed is from 75 to 90 per cent. of the weight of steel produced. Blister-steel may be used for steeling the faces of hammers and sledges, but its texture is not sufficiently uniform for general purposes; by re-heating and drawing, or by faggoting, welding, and hammering, or rolling, it is converted, in accordance with its degree of carburization, &c., into spring-steel or shear-steel. The former is produced by drawing out bars of mild blister-steel at a low heat, and the latter by making blister-steel into piles or faggots, re-heating in a hollow fire, and drawing into bars. The surfaces of these piles are covered with clay during the process of re-heating; this, by forming a vitreous slag, protects the combined or dissolved carbon from the action of the blast. After being once subjected to this treatment the product obtained is known as *single-shear;* by doubling the bars and repeating the process *double-shear* is produced.

*Crucible Cast-Steel.*—Although blister-steel may, by repeated workings under the hammer, be drawn into bars possessing tolerable uniformity of composition, yet this treatment is necessarily attended with a certain loss of carbon and consequent reduction of hardness. The requisite uniformity of structure may, however, be obtained by breaking up the crude bars obtained by cementation, and fusing the fragments in crucibles from which air is carefully excluded. The contents of these crucibles, when melted, are poured directly into cast-iron moulds, but where very large masses of cast-steel are required, a great number of crucibles are either emptied into a foundry-ladle before casting, or the pouring is so arranged that, by bringing up constant relays of fresh pots, a continuous stream of liquid metal is kept up. In this way large-sized ingots, 20 tons and upwards in weight, are currently made by Krupp of Essen, who employs crucibles containing 70 lbs. of steel. The material employed is stated to be a mixture of puddled steel and wrought-iron, with addition of carbonaceous matter; each furnace holds from 2 to 24 pots, and their removal is facilitated by a mechanical lifting apparatus placed below the ash-pit. The manufacture of cast-steel was introduced at Sheffield by Huntsman in 1740, and has been continued almost without modification to the present day.

The general arrangement of a steel melting-shop is exceedingly simple. The *melting-hole* or furnace is a rectangular cavity, from 18 inches to 2 feet square, and about 3 feet in depth to the grate, lined either with fire-brick or ganister. The top is on a level with the floor, the grate-bars and ash-pit being readily accessible from a cellar beneath; the cover is a square fire-tile, set in an iron framing with a projecting handle. A little below the mouth is a short rectangular flue, having a considerably less area than that of the furnace itself, and communicating with the stack, which, in order to command a sufficiently active draught, should not be less than 40 feet in height.

Several furnaces are arranged parallel with the walls on opposite sides of the melting-house, thus leaving in the centre of the floor a clear space for the moulds. The crucibles are made of a mixture of refractory clay from the Coal-measures, with ground potsherds and coke-dust, and are usually from 16 to 18 inches in height, and from 6 to 7 inches in diameter at the mouth. Two crucibles are generally placed in a furnace, the charge of each varying from 35 to 80 lbs. They are supported on discs of fire-clay standing on the grate-bars, but before being used require to be annealed by being gradually heated to redness in an open fire. This is done by placing them, in batches of about twenty, bottom upwards, together with their covers, on a layer of red-hot coal supported on a grate; the spaces between them are now filled with coke, and they gradually become heated to redness. The red-hot crucibles are removed to the melting-furnaces, and placed on their respective stands; the fires are replenished with coke, and as soon as the crucibles have become heated to redness, which takes place in about twenty minutes, they are charged with blister-steel alone or with other materials such as steel and wrought-iron scrap together, spiegeleisen or other manganiferous substances, according to the temper required in the finished product. This is broken into small pieces, properly assorted, and introduced through a wrought-iron funnel; after which the cover is placed upon the pot, and the full heat of the furnace kept up for four or five hours. A fresh addition of coke requires to be made about every three-quarters of an hour.

When the charge has become completely fused, which is ascertained by removing the cover and feeling the inside of the crucible with a long, pointed, iron rod, it is left at rest for a period which varies with the carbon temper of the metal. This operation, known as *killing*, is of the utmost importance for the production of sound ingots. The effect is probably due to the absorption of silicon, which is reduced from the substance of the crucible by the carburized molten metal. The surface of the metal is skimmed from any adhering slag, and the pot is lifted out of the furnace by means of tongs with strong concave jaws. The ingot-moulds, which are made of cast-iron, are often covered with a coating of carbon by being wiped with oil while still hot, or they may be washed with a mixture of clay and water, ground to the consistency of cream. As soon as the pot has been withdrawn from the furnace it is placed in the *teaming-hole*, which is a small pit in the floor, containing broken pieces of coke, where it is allowed to cool for a short time previous to pouring. When an ingot-mould has been filled, its mouth is covered either by a shovelful of dry sand or by a plate of sheet-iron.

When the first charge has been poured, the crucible, after being freed from any adhering slag, is returned to the furnace, in readiness for a second melting. The amount of metal now withdrawn is somewhat less than that melted during the first fusion; the time required for the operation is also less, and the consumption of coke proportionately diminished. The first melting occupies from four to five hours, while the second and third only require from two to two and a half hours each.

After from three to five successive meltings, the furnace is allowed to cool, since the very high temperature which would otherwise be attained would so corrode the surface of the lining as to greatly increase the area of the furnace, and thus cause waste of fuel. The total amount of fuel consumed is usually from three to three and a half times the weight of the ingots produced; but if the coke employed be of bad quality it may reach as high as five times the weight of the cast-steel made.

Where good coke is scarce small reverberatory furnaces, holding 9 pots, have sometimes been used instead of coke melting-holes, and Siemens' regenerative furnace has also been applied to steel-melting with considerable advantage. In the latter, the furnace, which has a removable arch, takes from 18 to 24 pots, and the fusion of 1 ton of ingots, instead of requiring 3½ tons of coke, is effected by the consumption of 1½ ton of ordinary coal-slack.

The fractured surfaces of ingots of cast-steel vary in appearance, in accordance with their hardness. The softer kinds are bright and finely granular, while the harder varieties exhibit distinct crystalline plates arranged in parallel bands at right angles to the surfaces of the mould, so that in a square ingot they exhibit a tendency to form a cross. The ingots produced are in all cases more or less unsound, containing vesicular cavities, which can only be removed by re-heating and hammering. This is effected at a low temperature, and access of air to the furnace during the operation is, as far as possible, prevented.

*Hindoo Process.*—According to Mr. Josiah Marshall Heath, *wootz*, or Indian steel, is prepared from iron made in the ordinary Hindoo furnace, by fusion in crucibles made of refractory clay, in which is placed, together with the metal to be converted, a certain portion of finely chopped wood, for which purpose that of the *Asclepias gigantea* or *Cassia auriculata* is preferred. The quantity of iron put into each crucible does not usually much exceed a pound in weight, and, after covering the pots with one or two green leaves of the *Convolvulus laurifolius*, they are closed with wetted clay, and placed in the sun to dry.

When the clay plugs have become sufficiently hardened, from twenty to twenty-four of these crucibles are built, in an arched form, on the bottom of a small blast-furnace, and strongly heated during from two to three hours with a blast produced by two bellows, each made of a bullock's hide. At the expiration of this time the conversion is considered to be completely effected; the furnace is then allowed to cool, and the crucibles are removed and severally broken, when the steel is found in the form of a rounded button occupying the bottom of each pot.

The cakes of steel thus obtained are prepared for drawing into bars by exposing them during several hours in a charcoal fire, to a temperature slightly below their melting point. The fire is urged by bellows, and the cakes are turned over before the blast; from this circumstance Mr. Heath arrives at the conclusion, that in order to insure complete fusion of the contents of the crucibles, the addition of a large excess of carbon is necessary, and that this excess in the too highly carburized steel is oxidized in the way above described.

The late Mr. T. H. Henry found that a specimen of wootz in the form of an inch square bar, four inches long, contained—

| C. Combined. | C. Uncombined. | Silicon. |
|---|---|---|
| 1·333 | 0·312 | 0·045 |

*Chenot's Process.*—The production of iron sponge by this process has been already described, and it therefore only remains to explain by what means the carburization of the metallic sponge is effected. For this purpose it is either intimately mixed with charcoal-powder or other solid material rich in carbon, such as a mixture of resin and charcoal, or it may be impregnated by imbibition with some substance rich in carbon, such as wood-tar or fatty matter. The ground sponge, after having been immersed in the carburizing liquid, is allowed to remain until complete saturation has taken place, and, when necessary, a gentle heat is employed for the purpose of facilitating imbibition. The metallic sponge, when thus saturated, is drained and torrefied in a close vessel during one hour, and when fatty matters have been employed as the carburizing agent, the sponge, after having been impregnated in lumps, is ground with the addition of 75 per cent. of fresh sponge to which no addition of carbon has been made. This is done to prevent over-carburization, and the production of too hard a steel. After having been thus prepared it is compressed into the form of small cylindrical masses occupying about two-thirds its original bulk, and these are melted in crucibles in exactly the same way as ordinary blister-steel. The siliceous and earthy ingredients of the ore form a slag which floats on the top of the molten steel, and, immediately before pouring, this is thickened by the addition of a little sand, and then removed by skimming. The charge of each crucible is from 18 to 25 kilos., and the operation occupies, on an average, four hours. Steel of fair quality has been produced in the way described, but the expense of fusion is necessarily great, since the compressed sponge occupies a much larger space than an equal weight of blister-steel, and consequently the charge of crucibles of the same capacity will be proportionately less.

*Mushet's Steel; Homogeneous Metal.*—A patent was granted in 1800 to David Mushet for a process for manufacturing cast-steel by fusing malleable iron in crucibles with a proper addition of carbonaceous matter. Different qualities of steel may be obtained by varying the proportion of carbon, a small quantity producing a softer variety than a larger one. The specification states that "steel produced with any proportion of charcoal not exceeding $\frac{1}{100}$ will generally be found to possess every property necessary to its being cast into those shapes which require great elasticity, strength, and solidity; it will also be found generally capable of sustaining a white heat, and of being welded like malleable iron; indeed, as the proportion of charcoal or other carbonaceous matter is reduced, the qualities of the steel will be found to approach nearer to those of common malleable iron." In his well-known "Papers on Iron and Steel" Mr. Mushet thus describes the properties of the metal produced: "When iron is presented in fusion to $\frac{1}{140}$ or $\frac{1}{150}$ of its weight of charcoal, the resulting product occupies a kind of middle state betwixt malleable iron and steel. It then welds with facility, and may be joined to iron or steel at a very high welding heat. Thus combined with carbon it is still susceptible of hardening a little, but without any great alteration in the fracture. It possesses an uncommon degree of strength and tenacity, and is capable of an exquisite degree of polish, arising from its complete solidity and the purity of fracture conveyed to it by fusion."

It will be observed that the process patented by Mr. Mushet is nearly identical with that by which wootz has, from ancient times, been prepared by the Hindoos. In 1839 a patent was granted to William Vickers for the production of cast-steel by melting 100 parts of iron borings with 3 parts of black oxide of manganese and 3 parts of ground charcoal. The use of scrap-iron is also claimed, the proportions specified being 28 lbs. of scrap, 2 lbs. 3 oz. of oxide of manganese, and 3 lbs. of charcoal.

*Case-hardening.*—This is a rapid process of cementation, by which the surface of wrought-iron may be converted into steel. An iron box is often employed as the cement-chest, and the charcoal used is, in most cases, obtained by charring some animal matter, such as horn, leather, or hoof. The objects to be case-hardened are imbedded in the charcoal in the usual way, and afterwards exposed for a short time to a moderate heat, either in a smith's forge or in some suitable furnace. When removed from the fire, the articles are hardened by being heated to redness and plunged, while still red hot, into cold water.

Small articles may be rapidly case-hardened by sprinkling a little ferrocyanide of potassium on their surfaces when red hot; as soon as the powder has disappeared the work is quenched in cold water in the usual way, and, if the process has been properly conducted, the portions covered by the salt will have become externally so hardened as to resist the file.

STEEL BY THE PARTIAL DECARBURIZATION OF CAST-IRON. *In Open Hearths.*—When, instead of causing carbon to combine in due proportion with malleable iron, steel is produced in an open hearth by the partial decarburization of cast-iron, the resulting product is known under the name of *raw-steel*, and may be employed for many purposes to which that obtained by cementation is commonly applied. This variety was formerly somewhat extensively manufactured on the continent of Europe, particularly in Styria and Carinthia; but the process has at the present time been almost entirely superseded by more improved methods. The crude iron best adapted for this purpose is that obtained from spathose ores and containing a considerable quantity of carbon, such as spiegeleisen, or the strongly mottled variety known as *blumige Floss*, which is speckled with grey upon a white ground.

After having filled the hearth with burning charcoal, six or seven plates or slabs of lamellar cast-iron are successively melted before the blast of the tuyer; these are from an inch to an inch and a half in thickness. At the commencement of the operation a certain quantity of rich slag and iron-scale, struck from the blooms by a large hammer, is added to the charge, which, melting on the surface of the cast-iron when in a fluid state, assists in the oxidation of the carbon which it contains.

When the first slab is in a liquefied state, and has collected at the bottom of the hearth, it is at first nearly fluid, but being there subjected to the oxidizing influences of the rich slags by which it is covered, it rapidly loses a portion of its carbon, and becomes thickened into a pasty mass. At this point another slab is fused by being brought directly before the blast, and this, falling in drops to the bottom of the hearth, again gives fluidity to the whole mass of metal there accumulated. Under the united influence of the blast and of the oxidizing slags, this in its turn loses a portion of its carbon, and becomes pasty. A third slab is now melted in the same way as the two former ones, but care is taken that the falling drops of liquid metal may be received on the centre only of the molten mass collected at the bottom of the hearth. The middle of the lump only is now melted by the fused cast-iron, this being surrounded by a ring of spongy metal which does not assume the liquid form. This operation is repeated until six or eight slabs have been successively melted, at the expiration of which time from 200 to 300 lbs. of spongy iron will have accumulated at the bottom of the furnace. The slags are, at this point of the operation, run off, and the metal is raised from amongst the fuel by which it is covered, and divided into wedge-shaped fragments by being cut according to a series of lines radiating from its centre to the circumference. By operating in this way, the several masses of crude metal will be found to have a nearly similar composition, but as the cake from which they are cut is itself far from homogeneous, the different parts of the same fragment seldom exhibit precisely the same degree of carburization. It consequently follows that these fragments, which are now drawn into bars, will yield rods of very different composition at different points of their length. To remedy this defect, and to give at the same time greater density to the finished steel, the bars of rough metal are handed over to a workman, who, after having heated them red hot, and subsequently cooled them by plunging in cold water, raises each bar by one of its ends, and allows it to fall heavily on an anvil placed for that purpose on the floor of the workshop. By this treatment the most brittle part of the bar is detached, and on striking a still harder blow in the same way, another and less carburized fragment is broken off, whilst the larger portion, which remains in his hands, merely consists of a peculiar steely iron, which, in some countries, is used for the teeth of harrows, for ploughshares, and for other agricultural implements.

The parts broken off are assorted according to the structure of the fractured ends, and are subjected to a series of manipulations destined to communicate to them greater density and uniformity of composition. For this purpose care is taken to weld together a piece of hard steel, and one which is less carburized; the bar thus obtained is afterwards heated, and hardened by being plunged into water, and this is again broken as before described, and subsequently united into one bar. It is easily understood that by this treatment the desired result will be ultimately attained; but this is produced at a large expense of labour and fuel, and is attended with the loss of a greater or less portion of the crude steel employed. The steel made by this process is, when carefully prepared, of excellent quality, and was at one time, for many purposes, preferred to that obtained by the cementation of malleable iron.

Forges of this kind are usually small, and worked by water-power. Each contains two fires and a hammer; a small water-wheel commonly gives motion to the bellows and another to the hammer. The latter weighs from 5 to 6 cwts., and makes from 65 to 110 blows per minute, with a lift of about 2 feet. Four men with the two fires will produce from 14 to 15 cwts. of rough steel blooms in a double-shift of sixteen hours; under ordinary circumstances the consumption of charcoal is about 30 cubic feet per cwt. of steel produced, but by using hot-blast, and placing a covering over the hearth, this may be reduced to about 22 cubic feet. The Carinthian process does not differ materially from the Styrian, but the hearth is larger, and the weight of metal operated on greater. In addition to performing the work of a refinery, the hearth, in this case, has also to do duty as a re-heating fire; the steel produced amounts to from 70 to 80 per cent. of the pig-iron operated on, and the consumption of charcoal is from 40 to 50 cubic feet per cwt. of steel produced.

In the Siegen district, where, before the introduction of puddling and other more recent processes for making steel, spiegeleisen was treated in the open hearth, small charges of from 60 to 80 lbs. were melted down upon a mass of mottled iron, which thus formed the bottom of the lump. The slag was tapped to within about 3 inches of the bottom, shortly after the commencement of fusion, and additions of spiegeleisen were made in diminishing quantities, from 40 lbs. at the fifth to 20 lbs. at the seventh and last charge. During these successive additions the mass was constantly maintained in a pasty semi-fluid condition, and, at the expiration of eight hours, a bloom, weighing 4 cwts., was obtained. This was divided into seven or eight pieces, which were tilted into bars, of which the weight amounted to about 70 per cent. of that of the pig-iron employed.

*Puddled Steel.*—Puddled steel appears to have been produced at Frantschach, in Carinthia, as long ago as 1835, but after repeated trials the process was ultimately abandoned. Bischof made puddled steel in a gas furnace at Mägdesprung in the Harz, in the year 1846, and during several years experimental trials were made by various ironmasters both in Westphalia and in Bavaria. In 1849 some of the Westphalian manufacturers had succeeded in overcoming all practical difficulties, and in the following year puddled steel had become an established article of commerce.

There is no essential difference between puddling for the production of wrought-iron and that for the production of steel, except that in the former case the decarburization is more completely effected than in the latter. The crude irons most suitable for conversion into steel are such as are rich in carbon and manganese, and consequently spiegeleisens, together with certain varieties of "mottled-pig," are well adapted for this purpose. Generally speaking, the furnace used is somewhat smaller than the ordinary iron-puddling furnace; or rather, the dimensions of the fireplace and chimney remaining the same, the size of the bed is somewhat diminished, in order, when required, to command a proportionately higher temperature. The charge is introduced in the form of fragments of nearly equal dimensions, and is so distributed over the surface of the bed that all may become fused about the same time and without the formation of any large quantity of oxide.

The charge of pig-iron does not commonly exceed from 3 to 3½ cwts.; in the preparation of puddled steel it is, however, necessary that the charge should not only be perfectly fused, but also that it should be covered by a stratum of liquid slag, which has the effect of regulating or rendering uniform the oxidation of the carbon. The presence of protoxide of maganese in the slag is likewise advantageous as contributing to its fluidity, without at the same time increasing its decarburizing influence.

The melting-down and stirring or rabbling is effected at a higher temperature than that employed when puddling for wrought-iron, and usually occupies from forty to forty-five minutes; the formation of the steel balls is, however, conducted at a lower temperature than those of wrought-iron, and at this stage of the operation the furnace should be filled with gases of a neutral or reducing character.

Fluxes of a more or less oxidizing nature, in accordance with the quality of the pig-iron under treatment, are added during the operation of melting-down; and towards the close of the process the presence of a poor, and therefore slightly oxidizing slag, in a state of great liquidity is required. The presence of viscid highly oxidizing slags would materially accelerate the fining, but would also be liable to result in a too complete decarburization of the metal, by which the quality of the steel would be prejudicially affected. The decarburizing action of the slags is regulated by the addition of clay, quartz, poor slags, mill-cinder, &c., as may be required. Sometimes addition of peroxide of manganese is made immediately before balling, or a mixture of peroxide of manganese, clay, and salt is added at intervals during the stirring. The contents of the furnace are well stirred during the second period of the process, and should the iron separated in a malleable form, accidentally become decarburized to too great an extent, it may be brought back to the proper condition by dissolving it in the still unaltered pig-iron beneath. When the metal commences to rise, the operation of fining is promoted by closing the damper until the charge begins to thicken, when the heat is gradually raised, and the mass is repeatedly worked with an iron tool; this stirring or rabbling occupies from forty-five to fifty minutes.

The appearance of the particles of metal, which are constantly brought to the surface of the covering of liquid slag by stirring, affords a tolerable indication of the progress of the operation and of the nature of the product which will be obtained. When the metal thus raised above the surface of the slag is brilliantly granular, it indicates that the process is progressing satisfactorily, and that the steel produced will be fine-grained and of good quality. If, on the contrary, the mass is coarsely granular and presents a flaky appearance, the steel will be likely to be coarse in texture and imperfectly refined.

The whole of the charge may either be balled-up at once, or a portion of it only, according to the nature of the steel required and the skill of the workmen employed; in some cases each ball is shingled as soon as it is finished, a new ball being formed in the meantime. In the puddling of iron a certain amount of decarburization takes place during the operation of balling; but in the case of steel this is, as far as possible, prevented by shutting the damper and filling the furnace with flame and smoke, thus producing a neutral or non-oxidizing atmosphere. When the furnace is heated by gas, the same result is obtained by shutting off the top blast. The shingling of the balls is conducted at a lower temperature than that employed for malleable iron, and those which cannot be immediately taken to the hammer are rolled in slag, so as to give them an external varnish, which tends to prevent oxidation. In order to prevent the decarburizing action of the slag, the balls should be shingled as quickly as possible; slags, when highly basic, act rapidly upon the combined carbon.

The fact of the partial decarburization of pig-iron requiring the expenditure of a larger amount of fuel than the more complete removal of its carbon, in the manufacture of wrought-iron, can only be explained by the slowness of the operation, caused by the peculiar circumstances under which the reactions are produced.

The time required, under ordinary circumstances, to work off a heat for wrought-iron and for steel will be, respectively, as follows:—

| | Iron. | Steel. |
|---|---|---|
| Melting-down . . | 30 to 40 minutes | 40 to 50 minutes. |
| Stirring . . . | 30 ,, 35 ,, | 45 ,, 50 ,, |
| Boiling and fusing . | 25 ,, 30 ,, | 20 ,, 25 ,, |
| Balling . . . | 10 | 10 |
| | 95 to 115 | 115 to 135 |

The consumption of fuel in puddling iron varies considerably, not only with its quality, but also with the nature of the metal originally operated on, and of that finally produced; it may, however, be taken roughly at from 100 to 125 per cent. of the puddled bars obtained. In puddling steel, however, from 130 to 135 per cent. of good round coal will be consumed; and, should the quality be indifferent, it may sometimes reach as high as 160 per cent.

The loss in puddling steel is less than that occurring in the production of wrought-iron. In puddling alone, it varies from 6 to 9 per cent.; if the loss on re-heating be included, it will amount to from 15 to 20 per cent.

From 1,800 to 2,000 lbs. of shingled steel balls can be produced from one furnace in twelve hours; when puddling wrought-iron from eight to nine charges are worked during that time, but with steel from six to seven charges only can be obtained.

The puddled balls, on being placed under the hammer, emit a blue flame, due to the combustion of carbonic oxide; and as they are less compact than those of wrought-iron, they require more careful management, first receiving very light blows and afterwards heavier ones. For the purpose of being drawn into bars, the blooms are re-heated, either in a reverberatory furnace or in a hollow fire. In large establishments puddled steel is generally re-heated in reverberatory furnaces, and subsequently treated by steam-hammers and rolling-mills; but in small works the re-heating is conducted in hollow fires, and the drawing-out is effected by the tilt-hammer.

This process, like many others previously noticed, though once of some importance, has now become almost insignificant owing to the development of the Bessemer and Siemens methods of steel-making on the large scale.

*Bessemer's Process.*—This process, the most important advance made in the metallurgy of iron since the introduction of hot-blast and the puddling-furnace, was first made public by Mr. (now Sir Henry) Bessemer at the Cheltenham meeting of the British Association in 1856, but some time elapsed before the conditions necessary for success on a working scale were determined, and more particularly the necessity of using only the purest kinds of pig-iron. Some of the earliest successes were obtained in Sweden, and subsequently pig-metal from Cumberland and Lancashire was almost exclusively employed in this country. This process essentially consists of blowing large quantities of atmospheric air, divided into numerous small jets, through a bath of molten cast-iron, thus effecting the rapid oxidation and consequent combustion of carbon, silicon, and certain other substances present in the pig-iron. The very high temperature which is thus developed in the converting-vessel is sufficient to keep liquid the resulting decarburized iron, instead of leaving it in the viscid pasty condition in which it is produced in the puddling-furnace.

The blast is injected at a pressure of from 18 to 20 lbs. per square inch, and the very high temperature attained is obviously the result of the intimate contact thus caused between the air and the various oxidizable bodies present. This oxidation takes place simultaneously throughout the whole mass, and not, as in the process of puddling, only at the surface, or where the metal comes in contact with cinder or some other oxidizing agent. The increase of temperature goes on progressively from the moment the blast is first turned on until it is again shut off; the various substances becoming oxidized in the same order of succession as they are respectively eliminated by refining and puddling. The silicon is thus first transformed into silica, which, uniting with oxides of iron and manganese, forms a liquid cinder. If the blowing be continued after the oxidation of the whole of the carbon has been effected, the heat will be kept up by the combustion of the iron itself, and a product is ultimately obtained which possesses all the properties of burnt iron. The blowing must therefore be stopped before this point is reached, or the metal must be "tempered" by the addition of appropriate carburizing materials.

By the first method, which was formerly employed in Sweden, the state of the charge, and consequently the period at which the blast should be discontinued, is determined by the appearance of the flame issuing from the converter. The results obtained by the second method, which was originally suggested by Mr. Mushet, are of a more certain and uniform character, and it is therefore now generally preferred.

Experience has everywhere shown that in order to obtain steel of good quality it is necessary to employ pig-iron of exceptional purity. It should in the first place be almost absolutely free from sulphur, phosphorus, and copper, as the process is, practically, incapable of reducing to any great extent the proportion of those bodies existing in the original pig-iron. On the other hand, the presence, within certain limits, of silicon and manganese is considered desirable, and, until the whole of the latter has been eliminated, oxidation of the iron takes place to a very limited extent; the silica resulting from the oxidation of silicon combines with manganous oxide and forms a liquid slag, which has, however, the disadvantage of exercising a corrosive action on the siliceous lining of the converter.

The English iron best adapted for use in the Bessemer converter is grey-pig smelted from Cumberland hæmatite, and of the quality indicated by the Nos. I. and II.; it should contain at least 1½ per cent. of silicon and not more than 0·2 per cent. of phosphorus. At Essen, Westphalia, the pig-iron preferred for treatment by the Bessemer process is smelted from a mixture of spathic ores and hæmatite; it contains 5 per cent. of carbon and 2 per cent. of silicon; it also averages 1 per cent. of manganese, 0·06 of phosphorus, and 0·04 of sulphur.

The furnace or *converter* employed in the production of Bessemer steel may either be stationary, like that formerly used in Sweden, or it may be suspended on trunnions, by means of which it can be rotated, vertically, through one-half or the whole of a revolution.

The fixed converter, which was at one time generally employed in Sweden, and which has been revived in a modified form by Clapp & Griffiths in South Wales, consists of a wrought-iron casing lined with fire-brick, somewhat like a foundry cupola, and provided on one side with a spout, by means of which it receives its charge of molten cast-iron. A series of refractory tuyers is placed in the bottom, and the whole is covered by a dome having an inclined hood, through which the gases evolved during the operation make their escape. The liquid metal is run into this converter after turning on the blast, so as to prevent the iron from filling the tuyer-holes, and the blowing is continued until the charge is run off by a tap-hole provided for that purpose.

The movable converter now generally employed, as shown in vertical section in fig. 108, affords great facilities for discharging the metal, and also allows of the charge being retained for a considerable time after the blast has been shut off. An external shell or casing, made of wrought-iron plates rivetted together, is suspended by means of a stout wrought-iron hoop, carrying trunnions supported by cast-iron standards. One of these is solid, and carries a pinion gearing into a rack on the extension of the piston-rod of a small direct-acting water-pressure engine; the other is hollow, and forms a passage for the blast. The lining requires to be composed of the most refractory material which can be obtained: fire-bricks are sometimes employed for this purpose, but, in this country, the fine-grained siliceous sandstone from below the Coal-measures, known as ganister, is found to answer better than any other material. It is first finely ground, and may be used either with or without an admixture of powdered fire-brick; in either case it must be intimately incorporated with a small quantity of water, by which it is rendered so far coherent as to retain its form when tightly rammed between the outside casing of wrought-iron and an inside wooden core, which is afterwards withdrawn. The older converters, of which the form was very nearly that of

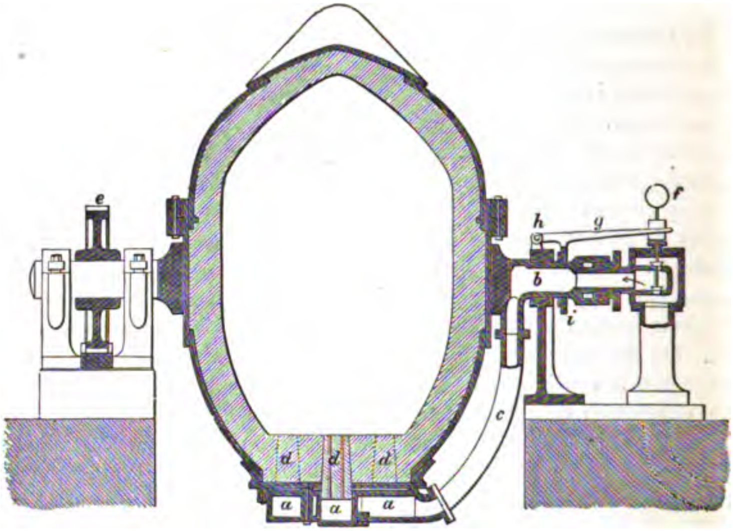

Fig. 108.—Bessemer Converter; vertical section.

a soda-water bottle with the bottom flattened and the neck turned on one side, were, for the convenience of lining, made in two parts, which were united by bolts and nuts. The form given to the newer converters is more cylindrical, and the bottom, which is removable, is retained in its place by cotter-bolts. Beneath the bottom of the converter (fig. 108, which is of a somewhat old pattern) is the tuyer-box, *a*, which is a cylindrical chamber communicating with the hollow trunnion, *b*, by means of the curved pipe, *c*. The tuyers, *d*, are slightly tapered conical fire-bricks, each perforated with from seven to twelve parallel holes. From five to seventeen of these tuyers are arranged in the bottom according to the size of the converter, at equal distances from each other. The lower ends pass through a perforated guard plate, forming the top of the air-chamber, with which they are maintained in close contact by stops supported by horizontal arms, which can be turned aside whenever the introduction of a new nozzle becomes necessary. The rack for turning the converter on its axis gears into the pinion, *e*, and, in the older establishments, the cylinders of the water-pressure engines were placed horizontally, but a vertical position is now more generally preferred, since less ground-space is occupied by this form of construction. The engine is in either case double-acting, and is worked by hand-gearing situated at a considerable distance. An arrangement for turning on and cutting off the blast by the rotation of the converter itself is shown in connection with the hollow trunnion, *b*; the valve, which is constructed on the double-beat principle, has its spindle prolonged through the top of the tubular pillar, and is so weighted at *f* as to keep it closed, when its fall is not mechanically interfered with. Attached to the valve-spindle is the lever, *g*, articulated at *h*, while to the trunnion is keyed the eccentric disc, *i*, which, pressing against the lever, lifts the valve and turns on the blast as soon as the apparatus is in a proper position for blowing. On the other hand, when the converter is lowered for the purpose of pouring, the pressure of the eccentric is taken off the lever, and the valve is closed. This arrangement for admitting and cutting off the blast at exactly the right moment, independently of any care or attention on the part of the workmen, was at one time generally used, but has since been abandoned.

In the small Bessemer plant, fig. 109, two converters, *a*, are placed on opposite sides of a circular casting-pit, in the centre of which is a vertical hydraulic cylinder, with a ram or plunger carrying a cross-arm, *b*, formed of two parallel iron girders rigidly connected by bolts and distance-pieces; to one end of this is attached the ladle, *c*, its weight being balanced by a counterpoise on the other end. This counterpoise is provided with gearing by which it can be gradually removed towards the centre in proportion as the ladle becomes emptied, and its weight consequently diminished. The ladle is made of wrought-iron lined with fire-clay, and is provided with a tapping-hole in the bottom, which is closed by the end of a bent iron bar also coated with clay; the other end of this bar turns downward on the outside of the ladle, and is connected

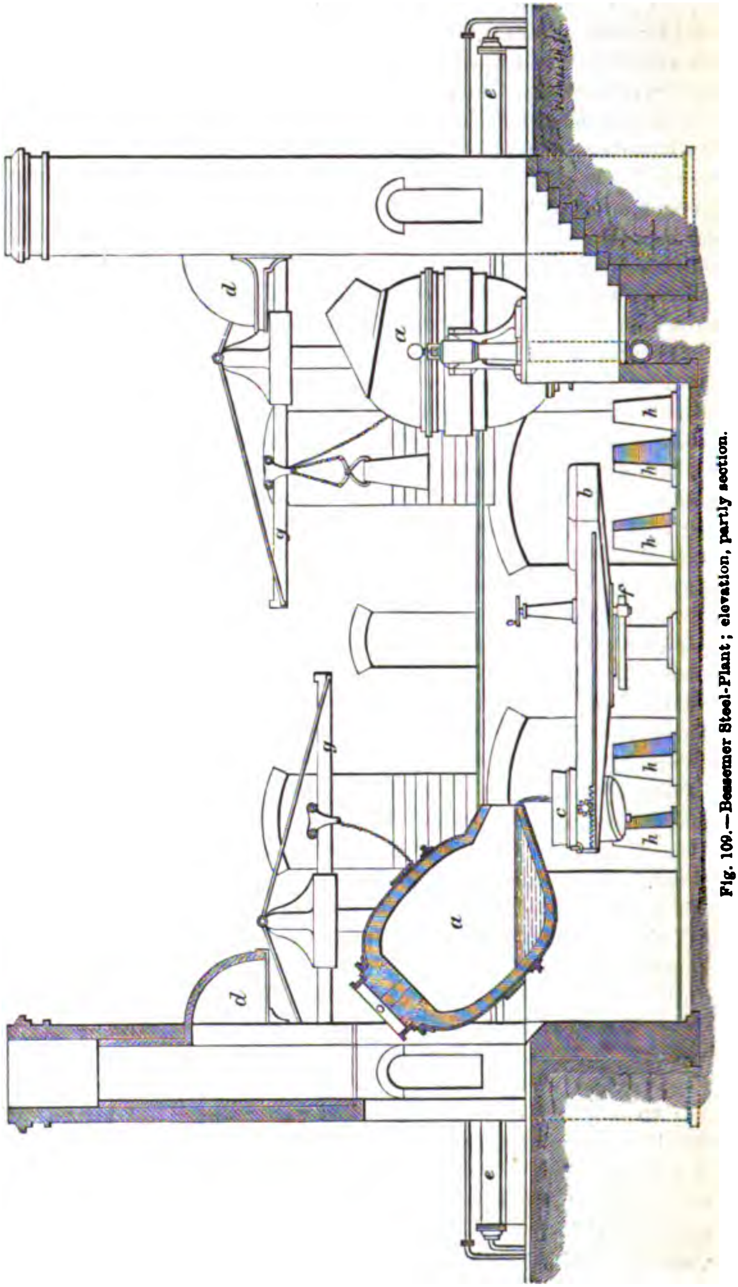

Fig. 109.—Bessemer Steel-Plant; elevation, partly section.

with a hand-lever, by which the plug in the tapping-hole may be raised or lowered at pleasure. The ingot-moulds are so arranged around the periphery of the casting-pit that the tapping-hole is brought successively over each, when the ladle and its support are revolved vertically on the central pivot; this motion is effected by means of spur-gearing, similar to that employed for the rotation of railway turn-tables.

This gearing is worked by a man from the crane-platform, who has also the control of machinery by which, after a cast, the ladle is turned over on its bearings for the purpose of removing any adhering cinder or waste. The valves regulating the admission of the blast, together with those for raising the ladle-crane and tipping the converters, are controlled by mean of levers from an elevated platform or pulpit, placed so as to command a distinct view of the operations, in which the workman in charge of the blowing is stationed. Power for the hydraulic apparatus is obtained from a steam-engine working force-pumps, in connection with a pair of accumulators. Each converter is usually capable of holding from 3 to 6 tons of pig-iron,[1] and, during the operation of blowing, occupies the position of that shown on the right hand, the flame and sparks being carried into a chimney by the hood, *d*. The hydraulic engines, *e*, are employed for tipping the converters, and a horizontal rack on the side of one of the girders is used as a slow-motion adjustment for bringing the tap-hole immediately over the centre of the mould. This is worked by a hand-wheel on the platform carrying the ladle. The rotary motion is obtained by a pinion gearing into a large spur-wheel, *f*, on the central plunger, and is worked by a wheel on the movable platform. The cranes, *g*, are worked by hydraulic power, and are employed for removing the ingots from the casting-pit.

The method of conducting the process in this country is generally as follows:—The charge is either re-melted in a cupola, or is tapped directly from the blast-furnace. In small works, as in Sweden and Styria, the converter may be placed below and in front of the furnaces; but more generally the latter are at a distance, and the metal is tapped into a large ladle mounted on wheels and drawn by a locomotive. The furnaces are usually raised in such cases so as to give room for a tunnel under the pig-beds in which the railway for the ladle runs; and the converter, which has been previously heated to redness by being filled with ignited coke, is first reversed, so as to remove any unconsumed fuel, and afterwards brought to a horizontal position to receive its charge of molten metal, which is run into it through an iron gutter lined with sand.

The converter is now slowly brought back to a vertical position, the blast being at the same time turned on; the flame which at first issues from the neck is of a yellowish-red colour, is but slightly luminous, and is not accompanied by a large amount of sparks. The reactions taking place at this period, which lasts from four to six minutes, are similar to those produced in the reverberatory furnace during the first stage of puddling. Graphitic carbon passes into the combined state, silicon becomes oxidized, and silicates of iron and manganese are formed. This stage of the operation is followed by a period of active ebullition, during which the combined carbon is rapidly oxidized by the blast, carbonic oxide is evolved in large quantities, the flame increases in brilliancy, and showers of sparks and fragments of burning iron are abundantly thrown out. This boiling period lasts for about six or eight minutes, at the expiration of which time the intensity of the action begins to diminish, fewer sparks are evolved, and the flame acquires a characteristic bluish-violet tint. This marks the commencement of the last, or fining, stage, and as soon as the whole of the carbon has been consumed, the flame ceases, but is immediately succeeded by a stream of white-hot gas, chiefly consisting of nitrogen; if, after this stage has been reached, the blowing be further continued, the temperature will be kept up at the expense of the decarburized iron, which becomes rapidly oxidized. As soon as the appearance of the flame indicates that the almost total removal of the carbon has been effected, the converter is again turned back to the horizontal position, and spiegeleisen, sufficient to give the required proportion of carbon in the finished metal, is added; this is run in from a furnace through a sand-lined gutter, in the same way that the charge was originally introduced.

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[1] Now increased to from 6 to 15 tons.

After the addition of the spiegeleisen it was formerly customary to again turn on the blast during a few minutes, but this is now discontinued, and the contents of the converter are at once emptied into the ladle; this is brought into the proper position to receive it by lowering the central plunger and moving it horizontally by means of the spur-and-pinion gearing before described. When very mild metal is made, the tempering is done with ferro-manganese, which is made hot and then shovelled into the ladle during the pouring. If the charge in the converter is very hot it is sometimes allowed to stand a short time before pouring, or crop-ends and similar pieces of scrap steel are added, which in melting down absorb the superfluous heat. The ingot-moulds, *h*, are of cast-iron, open at both ends; they have frequently a circular or octagonal section, and are somewhat smaller in diameter at top than at bottom. As soon as the ladle has been charged, it is raised sufficiently to clear the top of the moulds, which are arranged round the edge of the casting-pit, and is so turned as to bring the tapping-hole over the centre of each in succession; the plug is then lifted, and the mould beneath filled. All the other moulds are in turn filled in the same way, care being taken, in each case, not to allow the molten steel to impinge against the sides, since it is found that this is liable to result in the production of an unsound ingot. As soon as a mould has been filled, a small quantity of sand is sprinkled on the surface of the metal, which is then covered by a piece of sheet-iron, secured in its place by an iron cross-bar passing through eyes on either side. This plan of stoppering the ingot-moulds, which is also used in other processes of cast-steel making, is not always sufficient to ensure soundness in the ingots, especially when the metal is very mild or low in carbon, when it sometimes boils violently when poured, so that the mould cannot be filled. Other methods for treating the liquid steel in the moulds have been proposed, and to some extent adopted, with a view of obviating this inconvenience. These are noticed at p. 365. In charging, it is necessary that the blast should be admitted before the converter is turned so as to again assume a vertical position, since otherwise fused metal would flow back through the tuyers, where it would solidify and cause obstruction. The Bessemer process, when conducted in the manner described above, which is substantially the same as that originally developed by the inventor, is now known as the "*acid* process," in contradistinction to the so-called *basic* dephosphorizing method of Thomas and Gilchrist. These terms refer to the character of the material used for lining the converter, acid signifying quartz or ganister, and basic lime and magnesia. The analogy of the puddling process suggests the terms *dry* and *fluxing or boiling*, which might perhaps be more appropriately applied to the two modifications.

The nature and order of the chemical changes going on in the Bessemer converter have been investigated by several observers, and more particularly by Snelus, Kuppelwieser, Åkerman, and Müller, whose results show that slight but tolerably constant differences in the product may be traced to small initial differences in the composition and temperature of the charges, and the following classification of these methods is now used by German writers:—

1. Swedish method without complete decarburization.
2. English method for iron high in silicon.
3. German method for iron high in manganese.

The first method, now practically obsolete, is in some way represented by the following series of analyses, taken during the blowing of a charge at Neuberg in Styria; the metal, smelted from spathic ore with charcoal, was taken liquid from the blast-furnace and also used for re-carburizing:—

PIG-METAL AND PRODUCTS.

| | Grey Neuberg Cast-iron. | Metal taken after the Period of Scorification. | Metal taken towards the Close of Ebullition. | Burnt Iron taken before the Addition of Cast-iron. | Final Product. Mild Steel, No. 6. |
|---|---|---|---|---|---|
| C {graphitic . | 3·180 | ... | ... | ... | ... |
| C {combined . | 0·750 | 2·465 | 0·949 | 0·087 | 0·284 |
| Si . . . | 1·960 | ·443 | 0·112 | 0·028 | 0·033 |
| P . . . | 0·040 | 0·040 | 0 045 | 0·045 | 0·044 |
| S . . . | 0·018 | trace | trace | trace | trace |
| Mn . . . | 3·460 | 1·645 | 0·429 | 0·113 | 0·139 |
| Cu . . . | 0·085 | 0·091 | 0·095 | 0·120 | 0·105 |
| Fe . . . | 90·507 | 95·316 | 98·370 | 99·607 | 99·445 |
| | 100·000 | 100·000 | 100·000 | 100·000 | 100·000 |

CORRESPONDING SLAGS.

| | Slag from the Blast-furnace. | Slag taken after the Period of Scorification. | Slag taken towards the Close of Ebullition. | Slag taken before the Addition of Pig-iron. | Slag taken at the Moment of Casting. |
|---|---|---|---|---|---|
| $SiO_2$ . . . | 40·95 | 46·78 | 51·75 | 46·75 | 47·25 |
| $Al_2O_3$ . . | 8·70 | 4·65 | 2·98 | 2·80 | 3·45 |
| $FeO$ . . . | 0·60 | 6·78 | 5·50 | 16·86 | 15·43 |
| $MnO$ . . | 2·18 | 37·00 | 37·90 | 32·23 | 31·89 |
| $CaO$ . . . | 30·35 | 2·98 | 1·76 | 1·19 | 1·23 |
| $MgO$ . . | 16·32 | 1·53 | 0·45 | 0·52 | 0·61 |
| $K_2O$ . . . | 0·18 | trace | trace | trace | trace |
| $Na_2O$ . . | 0·14 | trace | trace | trace | trace |
| S . . . | 0·34 | 0·04 | trace | trace | trace |
| P . . . | 0·01 | 0·03 | 0·02 | 0·01 | 0·01 |
| | 99·77 | 99·79 | 100·36 | 100·36 | 99·87 |

When the metal is rich in silicon and poor in manganese its melting-point is high, and it is not very much superheated before blowing. These conditions are realized in the second or English method, and the first effect of the blast is the oxidation of silicon, as is seen in the following analyses of a blow made at Sheffield about 1870 :—

| | Original Pig Iron. | After Blowing | | End of Blow. | Spiegel added. |
|---|---|---|---|---|---|
| | | Six Minutes. | Twelve Minutes. | Eighteen Minutes. | Twenty Minutes. |
| C . . . . . | 3·570 | 3·940 | 1·640 | 0·190 | 0·370 |
| Si . . . . . | 2·260 | 0·950 | 0·470 | trace | trace |
| Mn . . . . . | 0·040 | trace | trace | trace | 0·540 |
| S . . . . . | 0·107 | 0·098 | 0·098 | 0·098 | 0·090 |
| P . . . . . | 0·073 | 0·070 | 0·070 | 0·070 | 0·056 |

In this case no carbon was removed until the bath had become well heated by the oxidation of silicon.

When the metal is poor in silicon, but contains a notable proportion of manganese, and is run into the converter in a strongly superheated state, the conditions of the so-called German Bessemer process are realised; the oxidation of carbon and silicon by the blast go on simultaneously, but the latter is not completely removed. The following is an example given by Müller for the Osnabrück Steel-Works, blowing a mixture of 70 per cent. of local and 30 per cent. of Cumberland pig-iron.

| | Charge. | After Blowing | | | Spiegel Added and Blown for Forty Seconds. |
|---|---|---|---|---|---|
| | | Five Minutes. | Ten Minutes. | Eighteen Minutes. | |
| C . . . . . | 3·460 | 2·710 | 1·630 | 0·092 | 0·104 |
| Si . . . . . | 1·930 | 1·070 | 0·790 | 0·532 | 0·346 |
| Mn . . . . . | 2·990 | 1·920 | 1·360 | 0·538 | 0·641 |

This method of working is more particularly adapted for the treatment of pig-iron with a notable quantity of phosphorus (0·10 to 0·15 per cent.), the effect of which is masked by the addition of silicon and manganese. Such steel, although under ordinary conditions fairly well fitted for rails, is, however, very brittle at low temperatures.

The first of the foregoing tables shows that copper and phosphorus are not oxidized in the Bessemer process, but that sulphur, when present in very small proportion only, finally disappears; silicon and manganese, more especially the former, are rapidly oxidized, while iron does not unite with oxygen, to any considerable extent, until silicon, manganese, and carbon have been almost entirely eliminated.

In Sweden nine grades of Bessemer steel are distinguished, according to their relative degrees of hardness, estimated by their tempering power. They are respectively designated by the numbers, 1, 1½, 2, 2½, &c., passing from the hardest to the softest. At the works of Siljanfors these various numbers were found to correspond very nearly with the following proportions of carbon:—

| | | |
|---|---|---|
| No. 1 . . . . . . | 2·00 | per cent. of carbon. |
| ,, 1½ . . . . . . | 1·75 | ,, ,, |
| ,, 2 . . . . . . | 1·50 | ,, ,, |
| ,, 2½ . . . . . . | 1·25 | ,, ,, |
| ,, 3 . . . . . . | 1·00 | ,, ,, |
| ,, 3½ . . . . . . | 0·75 | ,, ,, |
| ,, 4 . . . . . . | 0·50 | ,, ,, |
| ,, 4½ . . . . . . | 0·25 | ,, ,, |
| ,, 5 . . . . . . | 0·05 | ,, ,, |

No. 1 links white pig-metal with the hardest steel; it may be forged with difficulty but does not weld. No. 5, on the contrary, is homogeneous metal, welding perfectly but having no tempering power.

In Austria, where, as in Sweden, very pure pig-irons are treated by the Bessemer process, superior products are obtained. Tunner, the well-known metallurgist, has adopted a system of classification which differs but slightly from that employed in Sweden; he has, however, omitted the first two Swedish numbers, which rather belong to white pig-iron, and has replaced the half numbers by entire ones, from one to seven.

At the Imperial works at Neuberg, the proportions of carbon corresponding to the several numbers of hardness, are as follow:—

| Numbers of Hardness. | Proportions of Carbon. | Observations. |
|---|---|---|
| No. 1 | 1·58 to 1·38 per cent. | Cannot be welded, and is rarely used. |
| „ 2 | 1·38 „ 1·12 „ | |
| „ 3 | 1·12 „ 0·88 „ | Welds easily; used for bits, chisels, &c. |
| „ 4 | 0·88 „ 0·62 „ | Used for cutting-tools, files, &c. |
| „ 5 | 0·62 „ 0·38 „ | Mild steel, for tires, &c. |
| „ 6 | 0·38 „ 0·15 „ | Tempers slightly; steel for boiler-plates and axles. |
| „ 7 | 0·15 „ 0·05 „ | Does not temper; steel for pieces of machinery. |

It will be seen from these results that 0·25 per cent. of carbon, more or less, is sufficient to cause steel to pass from one grade to another. This is confirmatory of the theory which supposes, all other conditions being the same, that the hardness of steel will practically be proportionate to the amount of carbon it contains.

It has been pointed out by Jordan that a very large proportion of the heat developed by this process for manufacturing steel is due to the combustion of silicon, which, when converted into silica, combines with ferrous oxide and other bases, and covers the surface of the bath with a liquid slag; in the case of carbon, however, a considerable portion of the heat developed is abstracted by the carbonic oxide produced, which, escaping in the form of gas, is uselessly consumed at the mouth of the converter. He also states that in certain localities in the south of France the process can only be efficiently carried out by charging the converter directly from the blast-furnace, as the operation of re-melting, which usually results in the loss of 1 per cent. of silicon, so far reduces the proportion of that element as to render the resulting metal unsuitable for this method of treatment.

Manganese may, to a certain extent, replace silicon as a producer of heat, as in cases where the pig-iron operated on has been smelted from spathic ores. Silicon, although an essential component of good Bessemer pig, should not be present in excess, and, as a general rule, it should not exceed the amount of carbon in the iron. The presence of a very large quantity of silicon in pig-iron intended for treatment by the Bessemer process may be prejudicial to the result in two different ways; first, by giving rise to the formation of an increased amount of slag, resulting in a large loss of iron; secondly, by the difficulty experienced in accomplishing its complete removal by the time the elimination of the carbon has been effected.

The working of the Bessemer process is mainly controlled by the appearance of the flame, the sudden drop at the close of the decarburizing period, when blowing the hæmatite pig used in this country, being very marked; when much manganese is present the indications are more obscure, on account of the large amount of brown smoke which is emitted. The spectroscope has also been applied to the same purpose, the period of most active ebullition being marked by the appearance of bright lines and absorption bands, in the green part of the spectrum, characteristic of manganese and carbon respectively; these disappear when the flame drops.

Another method of controlling the operation is by the character of the slags. If an iron rod is plunged into the converter and removed at once, a portion of slag is brought out adhering to the point, which is of a peculiar brown tint as long as any carbon remains, but becomes dead black from magnetic oxide of iron as soon as it is entirely removed.

At Seraing the following relations have been observed between the colour of the slag and the amount of carbon in the metal:—

| | |
|---|---|
| Lemon-yellow | 0·75 per cent. and upwards. |
| Orange | 0·60 " " |
| Light-brown | 0·45 " " |
| Dark-brown | 0·30 " " |
| Bluish-black | 0·15 " " |

From a very excellent report to the Iron Office of Sweden, on the German, Austrian, and English manufacture of Bessemer steel, based on data collected in 1870, during a journey made in those countries by E. Brusewitz, published in the 'Jern-Kontorets Annaler' for 1871, p. 199, the following analyses are extracted:—

ANALYSES OF BESSEMER STEEL.

| | C. | Si. | Mn. | P. | S. |
|---|---|---|---|---|---|
| Steel made direct from the blast-furnace without addition of spiegeleisen: | | | | | |
| At Westanfors, Sweden | 0·085 | 0·008 | trace | 0·025 | trace |
| Ditto | 0·300 | 0·044 | 0·179 | 0·033 | trace |
| Ditto | 0·700 | 0·032 | 0·256 | ... | trace |
| Ditto | 0·950 | 0·047 | 0·463 | 0·032 | trace |
| Ditto | 1·050 | 0·067 | 0·355 | ... | trace |
| " Barrow-in-Furness (for coarse wire) | 0·200 | 0·179 | 0·214 | 0·026 | 0·030 |
| " Germany (for rail-heads) | 0·138 | 0·306 | 0·386 | 0·134 | 0·040 |
| " " (for rails) from iron poor in Manganese | 0·150 | 0·091 | 0·264 | 0·132 | 0·025 |
| " Germany (for rails) from mixture of Workington hæmatite pig with German manganiferous pig | 0·046 | 0·634 | 0·638 | 0·093 | 0·045 |
| " from Neuberg (for boiler-plate) direct from blast-furnace | 0·250 | 0·016 | 0·136 | ... | 0·010 |
| " from Neuberg (iron first re-melted in cupola) | 0·300 | 0·056 | 0·273 | 0·041 | 0·040 |

*Dephosphorizing in the Bessemer Converter.*—In the Bessemer process as originally conducted, it is necessary to employ pig-iron containing only a minute amount of phosphorus, as the proportion of that element removed during the conversion is very small. Such high-class metal is not only dearer but also less abundant than that smelted from oolitic and other cheap ores, and consequently the development of the process, although very considerable, was still restricted in one important direction. This original difficulty has now, however, been completely overcome by the adoption of the so-called basic method of Messrs. Thomas and Gilchrist, which was introduced in 1878. The principle upon which the process rests was discovered in 1872 by Snelus, who found that the retention of phosphorus in the blown metal was materially influenced by the character of the slag, which was essentially a silicate of iron and manganese of the form $RSiO_3$, and that when it could be reduced to a lower silicate by the addition of other bases, phosphorus could be removed by oxidation. Small trials made upon Cleveland iron at once proved successful, but the fact was not published, nor was the process adopted on the large scale, until more complete and elaborate experiments had been made independently by Thomas and Gilchrist.

The essential peculiarities of this process are the use of a converter lined with lime, magnesia, or with the combination of both as existing in calcined dolomite, and the addition of lime for the purpose of dephosphorizing after the carbon of the metal has been oxidized. This process is applicable to any variety of pig-metal that is sufficiently free from sulphur and does not contain more than about 1·0 per cent. of silicon, while phosphorus may be present in any reasonable quantity, and should not be less than 2 to 2½ per cent. Manganese in notable proportion is advantageous, as it prevents the absorption of sulphur in the blast-furnace, and by its oxidation provides a portion of the heat required to keep the metal fluid. The heat derived from the oxidation of phosphorus[1] is, weight for weight, about 25 per cent. less than that resulting from silicon. The condition of the carbon in the metal is not material, and white- is equally well suited with grey-pig if only the requisite freedom from sulphur be attained, as there is practically no desulphurization effected during the blow, in which respect the Thomas and Gilchrist resembles the original Bessemer process.

In some few instances the *converter linings* are made of bricks of caustic lime obtained from pure limestones, but more generally dolomite that has been exposed to a white heat for several hours is used. This is done either in cupolas or in regenerative chamber-kilns, the former being the simplest, although requiring rather more fuel than the latter. The cupola is built in sections, the central part being removable for convenience in re-lining. The lining is made of the same material (burnt dolomite) or of blocks of chromic iron ore. The latter, though an exceedingly refractory substance, is not very coherent, and great care is therefore required in cleaning out obstructions that may form during the working. The dolomite is charged in lumps of about the size of the fist; two barrows of coke are required for seven of dolomite. The burning lasts nine hours, and the stone comes out at a white heat. The fully burnt pieces when cooled are found to have contracted about one-half in dimensions, with a similar loss in weight. This so-called shrunk dolomite is very hard, and may be preserved in the air without change for some time, but it ultimately slakes and falls to pieces. From 12 to 15 cwts. of coke are required per ton of shrunk dolomite produced, and the cupola-linings last out from 40 to 60 charges.

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[1] P burnt to $P_2O_5$ = 5,760, Si to $SiO_2$ = 7,830, units per unit of weight.

The shrunk dolomite, when carefully freed from all imperfectly burnt or partially fused portions, is crushed under edge-rolls, in a Carr's disintegrator or in Vapart's mill, to the size of a pea, and mixed with from 3 to 12 per cent. of gas-tar that has been previously freed from water and ammonia by boiling. According to the amount of tar used the mixture is, when hot, either plastic or liquid. In the latter condition it is used for lining the converters by the method of ramming, while the stiffer mixture is used for making into bricks. These are moulded by stamping the mixture into iron moulds with hot iron rammers, and when filled the moulds are covered with cast-iron plates and heated for twelve hours in an oven, until the tar is burnt off. The bricks are moulded in segments to suit the converter, and each weighs about 55 lbs. when finished. They should be used as soon as possible, as under ordinary circumstances they disintegrate in from three to six days when exposed to the air.

One of the most modern forms of the Bessemer converter is seen in figs. 110, 111, in which it is shown in vertical section through the trunnions and in side elevation.

The converter is for an equal weight of charge larger than that of the original Bessemer process, as provision must be made for the very large volume of slag produced.[1] The woodcuts of p. 152 represent a new form of converter used at the Cleveland Steel-Works, which has a straight neck, and can be poured from either side. The outer shell or casing, made of wrought-iron plates 1 inch thick, is put together in three parts, namely, a central cylindrical body with a conical neck and bottom, so connected by eye-bolts and cotters that they may be detached from each other for repairs. The belt and trunnions are in two pieces, formed of cast-iron sections. The trunnions are 21 inches in diameter and 15 inches long; the belt is bored out to receive the body of the converter, which is secured by wedges, and can be removed by a crane over head. The total weight of the converter is about 40 tons, and of the belt 11 tons. The tipping-gear consists of a pair of double-acting hydraulic engines attached to one of the standards, working a steel screw of 4½ feet pitch and a worm-wheel 8 feet in diameter. This allows the converter to be turned to either side or completely round. When intended for 15-ton charges, the converters are 24½ feet high, and are mounted on piers or standards about 20 feet above the ground-level.

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[1] This requires an increased capacity of one-half, or a 6½-ton new converter must be as large as one of 10 tons for the original process.

The linings are either built up of dolomite bricks, moulded to the proper section, set in tar-asphalt, or with the plastic mixture of crushed shrunk dolomite and tar, which is rammed round a tapered iron core heated by a coke fire. The thickness of the lining is from 10 to 14 inches, the thickest portion being below; and when it is worn down to 3 or 4 inches, it is re-lined in place. The bottoms are made by ramming a mixture containing somewhat less tar than that used for the body-lining around a series of core-pins half-an-inch in diameter, which form the air-passages. The number of these varies from 35 to 80, according to their diameter, which may be from $\frac{4}{10}$ to $\frac{2}{3}$ of an inch. The thickness of the bottom is from 16 to 20 inches. When finished, it is burnt in an oven heated by two or more fire-places, in the same way as the bricks. In some cases, instead of using pin-tuyers larger cores are inserted, so that the ordinary siliceous tuyer-brick with numerous small air passages may be used. The body-linings originally lasted only from 40 to 50 blows, but now the number of blows has been increased to between 100 and 130. The bottoms last from 18 to 20 blows. When siliceous tuyer-bricks are used, if a proportion of them are renewed after every blow, they will last from 30 to 40 blows. When the converter is ready for charging, it is made hot by a coke fire, and a quantity of quicklime, from one-seventh to one-fifth of the weight of the charge of metal, is introduced. This is brought directly from the kiln, so as to be as hot as possible. The metal is then charged in the usual way, either from a cupola or directly from the blast-furnace. The blowing is conducted as

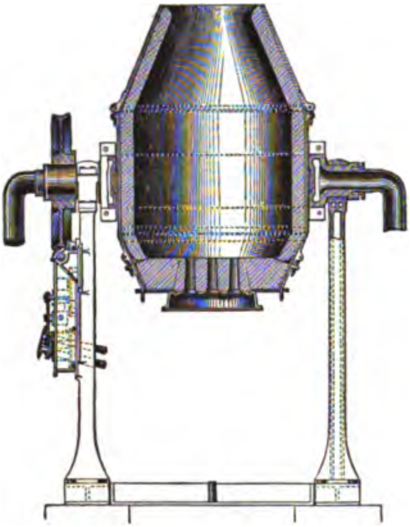

Fig. 110.—Vertical section.

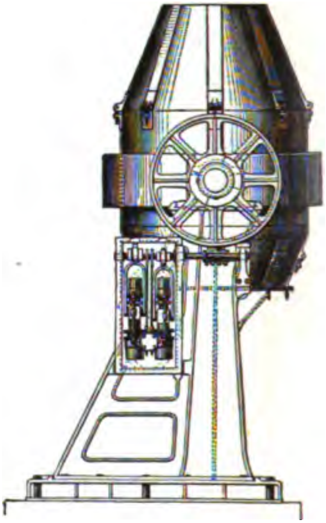

Fig. 111.—Side elevation.

Walker's 10-Ton Converter.

previously described. In about ten minutes the carbon is entirely burnt off, and the *after-blow* or dephosphorizing period commences. This is marked by a great increase in the temperature of the bath, and the thickening of the converter smoke from the burning of iron. The flame also shows peculiar bright patches, which are not observed in the old process. The length of the after-blow is regulated by breaking small test ingots, which should show a close silky fracture, entirely free from bright crystalline particles. When the desired result has been obtained, the converter is tipped to run off the slag, after which the metal is brought to the final temper by the addition of the proper quantity of spiegel and ferro-manganese, the former in the converter, and the latter in the ladle after pouring. The removal of the slag is of importance, as a portion of the phosphorus may be re-absorbed if left in contact with the metal for any length of time.

The re-carburizing may also be effected by adding a proportion of melted hæmatite pig to the over-blown metal in the ladle, which causes a strong ebullition with the production of slag which overflows the ladle. When this has subsided, spiegel or ferro-manganese is added as required before casting. The final pouring or teeming of the metal into the moulds is conducted in the usual way.

The chemical changes going on during the basic process have been very elaborately investigated both in this country and in Germany, and the results of these investigations, which are substantially similar, will be found in the later volumes (1880–83) of the *Journal of the Iron and Steel Institute.* The following table gives the results of the process as conducted at the Eston Steel-Works in Cleveland :—

METAL BEFORE AND AFTER BLOWING.

| | Original Pig-iron. | Six Minutes. | Twelve Minutes. | Fourteen and a half Minutes.[1] | Sixteen and a half Minutes. | Sixteen Minutes thirty-five Seconds.[2] | Finished Steel. |
|---|---|---|---|---|---|---|---|
| C . . . . . | 3·57 | 3·40 | 0·88 | 0·07 | trace | trace | 0·124 |
| Si . . . . . | 1·70 | 0·28 | 0·01 | trace | ... | ... | 0·030 |
| P . . . . . | 1·57 | 1·63 | 1·42 | 1·22 | 0·14 | 0·08 | 0·022 |
| Mn . . . . . | 0·71 | 0·56 | 0·27 | 0·12 | 0·10 | trace | 0·270 |
| S . . . . . | 0·06 | 0·06 | 0·05 | 0·05 | 0·05 | 0·05 | 0·040 |
| Corresponding slags— | | | | | | | |
| $SiO_2$ . . . . | ... | 42·60 | 35·60 | 33·00 | ... | 16·60 | 18·60 |
| $P_2O_5$ . . . . | ... | 0·15 | 2·61 | 5·66 | ... | 16·03 | 13·87 |
| Fe . . . . . | ... | 2·00 | 4·80 | 6·15 | ... | 11·35 | 7·10 |

The following analysis represents the average composition of the basic cinder produced at the North-Eastern Steel-Works in Cleveland :—

| CaO. | MgO. | $Al_2O_3$. | $Fe_2O_3$. | FeO. | MnO. | $V_2O_5$ | $SiO_2$. | $P_2O_5$. | $SO_3$. | Total. |
|---|---|---|---|---|---|---|---|---|---|---|
| 41·54 | 6·13 | 2·60 | 8·64 | 14·66 | 3·81 | 0·29 | 7·40 | 14·32 | 0·31 | 99·70 |

Metallic iron, 17·45. Metallic manganese, 2·95.

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[1] End of blow.

[2] End of after-blow.

The phosphorus in the slag appears to be mainly in combination with lime as tribasic phosphate. Sometimes tabular or columnar crystals separate in the slag, which are brown and slightly translucent by transmitted light, but tarnish to a black or blue tint; whose composition, according to Hilgenstock, is—

| CaO. | $P_2O_5$. | MgO. | MnO. | $SiO_2$. |
|---|---|---|---|---|
| 61·16 | 34·46 | 1·90 | 1·51 | 0·91 |

which corresponds approximately to a phosphate represented by the formula $Ca_4P_2O_9$ containing CaO 61·20 and $P_2O_5$ 38·14 per cent.

The loss upon the weight of pig-iron charged varies from 13 to 19 per cent., the average being 15 per cent., or about 3 per cent. more than with hæmatite pig. About 1 cwt. of basic lining-material is required for 2 cwts. of steel made, in the preparation of which $1\frac{1}{5}$ cwt. of coal or $\frac{4}{5}$ cwt. of coke is consumed. The lime used for fluxing is about 3 cwts. per ton of steel; and this, together with the wear of the lining, produces from 8 to 10 cwts. of cinder; a very much larger quantity than in the original process. Two principal applications have been suggested for this slag, namely, as a source of phosphorus in the blast-furnace, and as a fertilizer for agricultural purposes, the large amount of lime present being advantageous in both cases. When used in the blast-furnace, pig-iron may be readily produced containing 7 per cent. of phosphorus and upwards, although there is a considerable loss of unreduced phosphoric acid. For example, with 7·20 per cent. of phosphorus in the metal, 2·39 passed into the slag.

Phosphoric pig-iron may also be easily produced by smelting the tap-cinder of puddling-furnaces with Spanish manganiferous ores, in which way almost any desired proportion of manganese and phosphorus may be obtained in the metal.

The best way of using basic-cinder as a fertilizer is to grind it to as fine a powder as possible, and apply it to the land without further preparation. Various methods of preparing soluble phosphates from it have been tried by treatment with hydrochloric or sulphuric acid, but, hitherto, without success. When used in the raw state, it seems to be of greater fertilizing value than ground natural phosphates, and but little inferior to common superphosphates.

Of late years greatly increased rapidity of working has been attained in the Bessemer process by the use of methods that enable the worn parts of the converter to be replaced without cooling down. The most important of these is the loose bottom introduced by the late Mr. A. L. Holley in the United States, and now universally adopted. The bottom, which lasts, in the acid process, from 12 to 30 blows, according to the number of tuyer-bricks renewed during its life, and from 8 to 21 in the basic process, is moulded to a conical form, so as exactly to fit the corresponding enlargement in the body of the converter. When dried, it is brought from the stove upon a railway-truck, and lifted into place by a hydraulic lift, which may be either permanently fixed below the converter, or be portable and attached to the truck. In the latter case, the lift-pumps may be worked by hand, or power may be supplied from the main under pressure of the accumulator by a flexible tube. The connection between the bottom and the body is made by eye-bolts and cotters as shown in fig. 111. From 28 to 36 tons in twenty-four hours are now obtained in England from a pair of converters with ganister bottoms as compared with the 10 or 12 tons of early times; and in America from 45 to 75 tons are commonly realized.

The arrangements of the casting-pit have of late years undergone considerable modification in consequence of the larger charges worked and heavier ingots made. Instead of placing the converters at opposite ends of the diameter of a semicircular pit, they may be placed side by side, so that nearly the whole circumference of the sweep of the ladle-crane is available for moulds. Another method is to transfer the ladle when filled from the end of the crane at the front of the converter to another which commands the entire circumference of a second pit, which may be entirely filled with moulds.

When more than two converters are used in the same plant, they are usually placed with their trunnion-axes in line, and the ladle runs on a railway in front to receive the blown metal, which is then transferred to a casting-crane. This arrangement has been very extensively developed in the large open-hearth steel-works where a large number of furnaces are used. When the converters are placed high, as in figs. 110, 111, the ingot-pit is either very shallow, or the moulds may stand on the natural surface of the ground. This is an important advantage, as much better ventilation is secured than was possible in the old sunk pits. The newer forms, both of ladle and of ingot-crane, are made with two small hydraulic presses, in addition to the large central one. The latter is kept in continuous communication with the hydraulic ram under the accumulator, while the small ones are brought into use singly or together according to the weight to be lifted. In this way considerable economy in pumping-power is realized.

*The Uchatius Process.*—This process, which was patented in 1855, consists in effecting the partial decarburization of pig-iron by fusion in contact with ferric oxide or some other substance capable of yielding oxygen. The pig-iron is first granulated by running the fused metal into water, and the granulated cast-iron thus obtained subsequently mixed with about 20 per cent. of roasted spathic ore and 4 per cent. of fire-clay; this mixture is melted in clay crucibles in an ordinary cast-steel furnace. The softer kinds of welding cast-steel may be obtained by the addition of wrought-iron in small pieces to the above mixture, and the harder kinds by the addition of charcoal; the weight of the cast-steel obtained, when no addition of wrought-iron is made, is said to exceed that of the pig-iron by about 6 per cent.

The process has been tried in this country, and the principal objection to it seems to have been want of uniformity in the quality of the product obtained; it however appears to have been employed in Sweden, and samples of steel so produced were shown in the Swedish department of the International Exhibition of 1862.

STEEL BY FUSION OF A MIXTURE OF CAST-IRON AND WROUGHT-IRON.—The manufacture of steel by "reaction," in which wrought-iron is kept for a longer or shorter period immersed in a bath of molten cast-iron, has been long understood, and the result obtained appears to be partly due to the cementation of malleable iron at the expense of the carbon of the cast-iron, and partly to admixture of the two. Réaumur, in his treatise entitled 'L'Art de Convertir le Fer Forgé en Acier' (1722), says, "Iron is transformed into steel by immersing it for a short time in melted cast-iron," and adds, "This process for manufacturing steel is employed in some countries, and has been described by Vannuccio Biringuccio ('De la Pirotechnia,' lib. i. cap. 7, 1540)." Réaumur says in addition, that steel may be likewise obtained by fusing wrought-iron with cast, and that he had obtained steel in a common forge by thus mixing with cast-iron sometimes one-fourth and at others one-third of wrought-iron. He also states that cast-iron may be softened by *crocus Martis*, red oxide of iron. In 1798 Clouet states that iron or cast-steel may be obtained by melting pig-metal with oxide of iron.

Hassenfratz describes two furnaces which were used in England as early as 1812 for the manufacture of cast-steel by reaction:[1] "The mixture intended to produce steel is melted in ordinary reverberatory furnaces, in the lower part of which a kind of crucible is contrived. The metal, placed near the bridge, is heated, melts, and flows into the hearth, where it accumulates. The cast-iron becomes covered by slags, not only those contained in the iron, but also those formed by the partial fusion of the earthy glass that flows from the hearth. If the slag be in sufficient quantity, the bath is left at rest so long as the surface appears to bubble and carbonic oxide is disengaged and burns in the form of a strong flame. When ebullition ceases, a piece of green wood is introduced into the bath, and the liquid metal is stirred below the slag, in order to facilitate the separation of those scoriæ which remain in the cast-iron and adhere to the metal.

"At the moment the fining of the pig-iron commences, the principal workman introduces a small ladle into the bath and removes a little of the cast-iron from below the slags. He casts this into a test-ingot and tries it at the forge. He continues to take assays until what is taken out can be forged. Then he examines the grain of his steel; if it be too soft, he throws bars of over-cemented steel into the bath, to supply carbon without changing the mode of fining; if it be too hard, he throws into it clippings of wrought-iron, and sometimes even old iron, to dilute the carbon by increased volume, or to burn it partially; then he removes the scoriæ and pours into the mould the cast-steel, which is immediately forged into a commercial product."

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[1] Hassenfratz: 'Siderotechnie,' vol. iv. pp. 93–98. 1812.

*Obuchow's Steel Process.*—By this process white charcoal pig-iron of good quality is re-melted in a cupola-furnace and tapped into a large crucible, previously heated to bright redness, containing malleable iron or steel scrap, together with magnetic iron ore, titaniferous black sand and clay; arsenious oxide and nitre are subsequently added. In some cases addition is made of magnetite and arsenious oxide only. After receiving the charge of molten pig-iron, the crucible is heated until its contents have become perfectly liquid, when the nitre and arsenious oxide are added, and the whole is well stirred.

The steel is cast in vertical cast-iron moulds, and, when sufficiently cold, is drawn out under tilt-hammers. The proportion in which the ingredients are mixed must manifestly exert an essential influence on the quality of the product obtained.

The following analyses of Obuchow's common steel are by Chodnew:—

| | | |
|---|---|---|
| C graphitic | 0·15 | 1·25 |
| C combined | 1·02 | |
| Si | 0·04 | trace |
| Fe | 98·79 | 98·75 |
| | 100·00 | 100·00 |

This steel is said to be principally employed in the manufacture of guns, cannon, &c.

*Price and Nicholson's Process.*—A patent was granted in 1855 to David Simpson Price and Edward Chambers Nicholson for a method of manufacturing cast-steel by melting together malleable iron and refined metal—that is, pig-iron freed from the chief portion of its silicon; the relative proportions of cast- and wrought-iron are to be adjusted in accordance with the nature of the cast-steel it is desired to produce. Shortly afterwards Mr. Gentle Brown obtained a patent for the manufacture of cast-steel by fusing bar-iron with good charcoal pig. A patent was granted in 1862 to Charles Attwood for producing the same result by similar means.

*Open-Hearth or Siemens-Martin Process.*—The production in the reverberatory furnace of cast-steel by the solution of malleable scrap in molten pig-iron, in accordance with the method proposed by Price and Nicholson, Brown, and others, has been brought to a considerable degree of perfection by the use of the Siemens regenerative gas furnace. The first experiments carried out on a working scale were made by M. Martin at Sireuil, near Paris, in 1865; but the practical success of the process appears to have been mainly due to the adoption of the gas furnace of the Messrs. Siemens.

The furnace, which is generally similar to that shown in figs. 25, 26, 27, pp. 104, 105, has only a single door, which is in the middle of one of its longer sides, while on the opposite one, and at the lowest part of the hearth, is a tapping-hole and a channel, through which the metal is conducted for casting; the horizontal section is a rectangle with the corners removed. The hearth is composed of refractory sand, supported on an iron bottom kept cool by a current of air, and is repaired after each operation by ramming fresh sand into any holes which may have been produced. On the casting-side of the furnace an iron tramway with waggons, or a revolving platform, brings the ingot-moulds successively under the tap-hole until the casting is finished. Alongside the melting-furnace is an ordinary reverberatory furnace with a flat hearth, in which the pig-iron and packets of scrap added during the operation are heated to redness. At Sireuil the pig-iron employed was principally obtained from the blast-furnaces of St. Louis, near Marseilles, and from Ria, near Prades; the charge was from 1,500 to 2,000 kilos. (3,300 to 4,400 lbs.); but in the large modern furnaces charges up to 20 tons are currently worked. The furnace, after being heated to whiteness by gas passing through a Siemens regenerator, is first charged with a certain weight of pig-iron, which, to prevent chilling the furnace, is previously heated to redness in the auxiliary furnace before mentioned. When the pig-iron has become melted and the bath is very hot, wrought-iron is added in quantities of from 10 to 20 kilos. at a time. Additions of red-hot iron are made at intervals of from twenty to thirty minutes, each addition being followed by a vigorous stirring, in order that the wrought-iron may be more readily dissolved and more thoroughly disseminated in the bath.

With a neutral flame a bath of No. I. grey-iron will dissolve 10 times its weight of Bessemer steel scrap, containing 0·3–0·4 per cent. of carbon; but No. III. metal will not take more than three or four times its weight of puddled iron, and if the flame is oxidizing, considerably less.

When the assays taken show that the metal has been sufficiently fined, pig-iron heated to redness is charged in place of wrought-iron or iron ore, and, after the whole has been thoroughly stirred, another sample is taken, which determines, approximately, the further amount of pig-iron to be added. After two or three successive additions have been thus made, samples are withdrawn every half-hour until metal of the proper quality is obtained, when the charge is tapped into ingot-moulds.

Each operation occupies from nine to eleven hours, with 5 to 6-ton charges.

The loss when melting Bessemer scrap with No. III. hæmatite pig is from 4 to 5 per cent. of the total weight charged. The coal consumed is from 13 to 14 cwts. per ton of ingots.

*Newer Forms of Open-Hearth Furnaces.*—Since the original introduction of the Siemens process many modifications have been made, both in the size and shape of the furnace, for the different purposes of accelerating the operation, increasing production, and facilitating repairs. For a long time the furnace was somewhat of the character represented in fig. 25, &c.; subsequently the air- and gas-passages were made longer and narrower, and the mouth of the former was placed at a higher level than the latter, so as to promote combustion by the rapid intermixture of the two currents, the specifically lighter gas rising through the denser air above it. The regenerators have also been protected by the introduction of semicircular arched roofs instead of the flattened segmental form at first employed.

Another modification is the Pernot rotating furnace, introduced at St. Chamond, France, in 1874, represented in transverse section in fig. 112. This has a circular bed in a casing of cast-iron plates built up like a cask, hooped with wrought-iron, and mounted upon an axis inclined about 6° from the vertical, by which it may be rotated by appropriate mechanism. The whole arrangement is mounted upon wheels running upon rails, so that it may be withdrawn for repairs as shown in dotted outline on the left side of fig. 112. When in working position, the carriage with the hearth stands upon the top of the regenerator, and under the arch of the furnace. When at work, the bed makes from two to four rotations per minute; the pig-iron and scrap are heated to redness before charging, and as the position of the pieces is continuously changing from full exposure to the flame to immersion in the liquid bath, the fusion is very rapid. The bottom of the hearth being also exposed to the flames at short intervals, is kept at a strong heat, so that chilling or sticking of the charge to the bottom is prevented. The diameter of the hearth is about 7 feet in an 8-ton furnace, and 13½ to 15 feet in those for 20 and 25 ton charges.

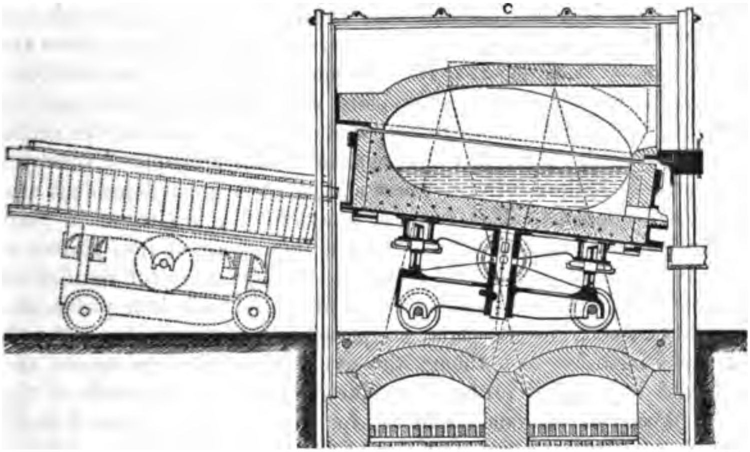

Fig. 112.—Pernot's Open-Hearth Furnace; transverse section.

As an example of the working of this furnace, the details of the progress of the operation of melting a 20-ton charge are given on the next page.

| Time. | Pig-Iron. | Steel Scrap. | Puddled Iron. | Ferro-Manganese. | Carbon in Metal. |
|---|---|---|---|---|---|
| | Tons. Cwts. | Tons. | Tons. | Cwts. | Per cent. |
| 2.10 A.M. | 5 0 | 5 | ... | ... | ... |
| 4.45 A.M. | ... | 3 | ... | ... | 0·5 |
| 7.0 A.M. | ... | 3 | ... | ... | 0·4 |
| 9.40 A.M. | 0 10 | ... | 2 | ... | 0·25 |
| 11.50 A.M. | ... | ... | 1 | ... | 0·25 |
| 1.25 P.M. | ... | ... | ... | 13¼ | 0·12 |
| 1.30 P.M.<br>Metal tapped out. | ... | ... | ... | ... | 0·35 |
| | 5 10 | 11 | 3 | 13¼ | ... |

The coal used was at the rate of about 6¾ cwts. per ton of ingots made, the loss on the materials being from 5 to 7 per cent., according to the temper of the metal, being the largest for the mildest qualities. The saving of fuel is about one-half that required by the fixed-hearth furnace. The original plan of building the furnaces immediately above the regenerators has been departed from in many cases. At Gratz in Styria the furnaces are built upon brick piers, whose height is determined by the requirements of the casting-ladle, while the regenerators are in vaults below the ground. A further development of this principle of construction is seen in Batho's furnace, where all the parts are independent and accessible for repairs. This furnace, like Pernot's, is circular in plan, but the bed is fixed, and carried upon a framework of iron girders at such a height above the ground as to give a large space for ventilation. The regenerators, also of circular form, are cased with iron plates, and resemble Cowper's stoves. They are placed at the corners of a rectangle with the hearth between them; the gas and air-tubes are square, and cased with iron. This furnace, introduced in 1884 at the works of the Steel Company of Scotland, is in use for charges varying from 3 to 15 tons. In the largest, the hearth is 13 feet and the regenerators 6½ feet in diameter; the latter being 12 feet high.

Mr. Frederick Siemens has lately modified the original open-hearth furnace by substituting a cylindrical or domed roof for the flattened form, depressed in the middle, previously in general use. In the latter construction the flames are directed by the use of sharply inclined ports upon the centre of the hearth, so as to communicate heat to the charge by contact; while in the former, the raised roof and horizontal ports give an enlarged chamber above the bed, in which the gas burns out of direct contact with either the roof above or the charge below, the work being done by radiation from the freely developed flame. Furnaces whose roofs have been altered to this form are said to work with a considerable saving of fuel, and the roof, being protected from the cutting influence of the flames, is more durable.

*Basic Open-Hearth Process.*—The Thomas and Gilchrist process of dephosphorization has also been successfully applied to the manufacture of steel in the open-hearth furnace, a bed of shrunk dolomite or magnesia being substituted for that of ganister used in the Siemens process. Figs.

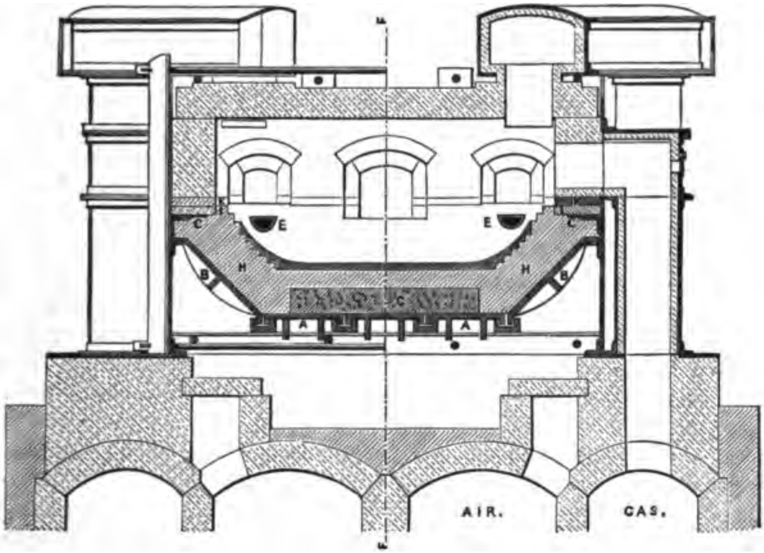

Fig. 113.—Gillot's Basic Open-Hearth Furnace; vertical section.

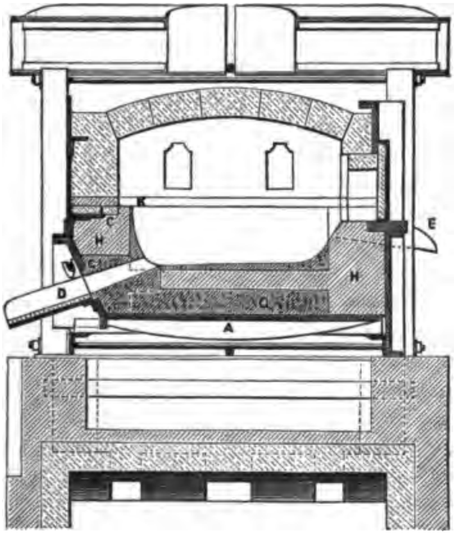

Fig. 114.—Gillot's Basic Open-Hearth Furnace; transverse section on F F.

113, 114, represent a new form of furnace for this process by Mr. Thomas

Gillot[1] at Farnley, near Leeds. The iron bottom-plates, A, and bridge-plates, B, are arranged in the usual way above the regenerators, while the superstructure of siliceous brickwork is carried on brackets, C, bolted to the side-plates of the furnace. This prevents irregular settlement in the event of the bars in the hearth shrinking or becoming fluxed. The tap-hole and spout for the finished steel are in front of the furnace at D, on the centre line of the bed, while the two slag-holes, E, together with the charging doors, are placed near the bridges at a higher level on the opposite side. The bottom, G, is made of shrunk dolomite and tar mixture rammed with red-hot rammers, the passages for the tap-holes being preserved by wooden plugs. The parts marked H are built with dolomite bricks grouted with the same mixture as is used for ramming. Two isolating courses, I and K, supported upon brackets extending round the sides and ends of the furnace, keep the basic part of the bed out of contact with the siliceous brickwork of the roof and gas-passages; the lower one, I, is of magnesia, the upper one, K, of bauxite bricks.[2] The shape of the hearth is finished with lime rammed above the brickwork. The gas- and air-ports, instead of being included in the body of the furnace, are enclosed in separate casings outside the hearth, so that they may be changed and the inlets repaired without cooling the furnace. This arrangement of gas- and air-ports, by which the two currents are introduced at different levels instead of side by side, is due to Messrs. Hackney & Wailes. The furnace, when newly built and fired, will be sufficiently hot in seventy-two hours for forming the bottom, for which purpose a full melting heat is necessary. Well-burnt and partly ground dolomite mixed with at most 10 per cent. of ground fire-brick is added, so as just to frit the lining and bind the layer forming the bottom. The wooden tap-hole plugs are burnt out and the passages stopped by dry lime alone, or with lime mixed with coke-dust. The operation can then be commenced by laying upon the bottom a quantity of quicklime, from 5 to 10 per cent. of the weight of the pig-iron, which is then charged upon it, and the wrought-iron or steel scrap over both, all cold. The whole charge should be introduced at once, and feeding afterwards as much as possible avoided. In three or four hours, when the pig has melted and the charge somewhat sunk, the slag, which is generally fluid and siliceous, should be removed, in order to protect the sides of the hearth from undue corrosion. In from three and a half to six hours, according to the proportion between the pig and scrap used, the whole charge will be melted, when more slag can be taken off or lime added if it is too fluid. When the proportion of pig is high, iron ore may be added with the lime. When the boiling has ceased and the fluid metal has been well stirred, it is sampled and tested as in the basic-converter process. And when the result is found satisfactory, the slag is again removed; but it is now too pasty to run, and must be raked off through the centre charging-door, which is 4½ inches lower than the end ones. About 5 per cent. of hæmatite pig-iron is then charged on to the bridges, which, when melted, runs down into the bath, causing a violent boiling for about fifteen minutes. The furnace is then allowed to rest for a quarter or half an hour, and any slag more than is necessary to protect the metal in the ladle is removed, and the bath is then ready for the addition of spiegel or ferro-manganese and tapping. This addition is preferably made in the furnace rather than in the ladle, as the action is very violent, and a more uniform mixture is secured. The slag remaining in the furnace after casting is raked out at once, and the tap-hole is cleared from the opposite door. The bottom and sides of the furnace basin are then repaired with finely ground calcined dolomite, and gas is turned on to bring it to a state of semi-fusion, which requires from one to three hours, and the furnace is then ready for the next charge.

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[1] 'Proceedings of the Institution of Civil Engineers,' vol. lxxvii. p. 297.

[2] At the Alexandrowsky Works, St. Petersburg, an isolating layer 6 inches thick of chromic iron ore mixed with tar was interposed between the siliceous bricks and the basic hearth.

The process above described is adapted for treating best Yorkshire cold-blast pig-iron, containing silicon 1·245 per cent. and phosphorus 0·601 per cent.; the scrap iron averages 0·2 per cent. of phosphorus, which is reduced in the finished ingots to between 0·055 and 0·097 per cent. The following are the details of one operation:—

| | | | |
|---|---|---|---|
| Charged | 6.30 A.M. | Farnley No. 3 pig . . . | 28 cwts. |
| | | Steel scrap . . . . | 8½ ,, |
| ,, | 2.45 P.M. | Hæmatite No. 3 pig . . | 2 ,, |
| ,, | 3.10 ,, | Spiegel, 18 per cent. Mn. . | 1¼ ,, |
| ,, | 3·25 ,, | Ferro-manganese, 53 per cent. Mn. | ½ cwt. 5 lbs. |
| | | | 40¼ cwts. 5 lbs. |

At 10.30 A.M. 4½ cwts. Somorrostro ore and 1¾ cwts. of burnt lime were added; 2 cwts. of slag run off at 2.20 P.M., and 3½ cwt. after sampling. The cast at 3.28 P.M. gave one 15-inch and one 12-inch square ingot, which, after hammering, were rolled into plates.

The average yield of ingots is about 93 per cent. on the total weight of pig-metal, scrap, and ferro-manganese charged. The dolomite used in repairs is about 9 cwts. per ton of ingots. The addition of lime during the melting is very destructive to the gas and air ports, as small quantities are carried over by the draught and flux the siliceous brickwork, forming a slag which, dropping on to the bridge-plates, causes a further fluxing of the lime. The ports usually require repairs after from eighteen to twenty-one days' work, and the regenerators require to be cleaned from the deposit caused by the brown smoke in the later stages of the process, after sixty or seventy casts have been made. The quality of steel produced being very soft, 0·12 to 0·24 carbon, the heat is near the limit of endurance of ganister bricks, and, but for their great cost, magnesia bricks would probably be preferable for facing the gas-ports.

Magnesia for converter- and furnace-lining may be obtained by calcining magnesite or native carbonate of magnesium, which is, however, a somewhat rare mineral, and apt to contain silica, or from dolomite by the removal of the lime. For this latter purpose the mixed lime and magnesia obtained by calcining dolomite may be treated according to Scheibler's method by sugar in the form of molasses or syrup, which gives a saccharate of lime soluble in water, and hydrated magnesia. The latter being insoluble, precipitates, and can be separated by a filter-press. The saccharate of lime is then decomposed by passing carbonic acid through the filtered solution, which precipitates carbonate of calcium, leaving the sugar in a condition to be used over again. The magnesia obtained by this method at Hörde contained only about 2 per cent. of lime and 1·0 per cent. of silica. It was plastic and could be easily moulded.

Closson's method was intended for use with the waste liquor of the Stassfurt Potash-Works, which contains about 15 per cent. of $MgCl_2$. When this is heated with burnt dolomite, chloride of calcium and magnesia are formed, thus—

$$MgCl_2 + CaOMgO = CaCl_2 + 2MgO.$$

The operation is effected by agitating the magnesium chloride liquor with the burnt dolomite diffused as a "milk" through water in a tub with rotatory stirrers. The decomposition takes place rapidly, and when completed the contents of the tub are put through a filter-press. The calcium chloride runs out, leaving hydrate of magnesium in the filter, which when washed with water is practically pure. The liquors at Hörde, where the process was tried, were further utilized by mixing them with a fresh portion of burnt dolomite in water, and passing blast-furnace gas through the mixture, which was kept well stirred to promote contact between the gas and the liquid. Carbonate of calcium is precipitated by the carbonic acid in the gas, which is thereby improved for heating purposes, while magnesium chloride is re-formed, thus—

$$CaCl_2 + MgOCaO + 2CO_2 = MgCl_2 + 2CaCO_3.$$

The liquor containing the chloride of magnesium is drawn off, filtered, and used again. The mud and calcium carbonate separated are thrown away, and the loss of magnesium chloride is from 5 to 6 per cent.

On the French coast of the Mediterranean, at Aigues-Mortes, Closson's process is applied to the recovery of magnesia from sea-water, which contains magnesium chloride and sulphate to the amount of 4 lbs. per cubic yard. The sea-water is pumped into a tank lined with masonry, and milk of lime in the proportion of 1·5 units of lime per unit of magnesia is run in simultaneously. The mixture is made by mechanical stirring in two similar tanks, and is filtered into shallow pans 16 feet wide and 1,000 feet long, with beds of clean beach-sand. The precipitate is allowed to dry in the sun, which requires from twenty to thirty days in summer, but in winter artificial heat is necessary, and when dried the hydrated magnesia is calcined and made into bricks with tar in the usual way.

*Compressed Steel.*—In casting steel in large masses, whether from crucibles, Bessemer converters, or open-hearth furnaces, it is often difficult to obtain sound ingots, especially when the metal is of a very mild temper, the upper part being spongy from included hollow spaces, and sometimes perfectly honeycombed by the arrangement of these holes in lines perpendicular to the surfaces of the mould. This effect has been attributed either to the escape of carbonic oxide formed by the reaction of carbon in spiegeleisen upon oxidized iron when the final tempering is effected, or to the escape during solidification of hydrogen previously absorbed by the molten metal. Although neither of these views has been fully established by experiment, it is certain that iron and steel when solidified retain a considerable quantity of gas, which may be extracted by heating the metal *in vacuo*, and consists principally of hydrogen (65 to 90 per cent.), and nitrogen (30 to 10 per cent.), with at most 2½ per cent. of carbonic oxide.

The consolidation of ingots intended for rolling or forging has been effected by subjecting the metal while in the mould to hydrostatic or gaseous pressure until it sets, whereby the escape of dissolved gas is prevented, and also by stirring, and transferring from one ladle to another to give facility for such gas to escape. The first method is applied on the largest scale by Sir Joseph Whitworth in the production of so-called compressed steel, which is subjected while melted to a pressure varying from 6 to 20 tons per square inch, the press used being capable of exerting a strain of 10,000 tons. The ingot-moulds are heavily hooped with steel, and lined internally with fire-brick. The ingots are said to be about 12½ per cent. shorter than those cast in open moulds, but the density of the metal is only increased by about $\frac{3}{10}$ths per cent. (from 7·85 to 7·88 at 15° C.) Other methods of applying pressure are those of Captain Jones of Pittsburg, using steam, and Baron Krupp, using liquid carbonic acid. In the former, the mould is clamped firmly to its base, and a conical seat is turned on the neck, which, when the metal is poured, is covered by a closed lid wedged down, so as to form a steam-tight joint. The top of the lid communicates by a flexible pipe with a drum supplying steam at a pressure of from 80 to 150 lbs. per square inch, which is allowed to act until the metal has set. This plan is said to have given good results, but is believed to be no longer in use.

In Krupp's method liquefied carbonic acid, which is now made commercially on a large scale, is used instead of steam. The bottle containing the liquid acid communicates with the upper part of the mould by a pipe of small bore. The mould is hooped with steel; the cover is secured by bolts and wedges, and an expanding copper ring forms a gas-tight joint. The cover is bored through to allow the introduction of the liquid metal, and the passage can be closed by a sliding wedge. When the mould is filled, the surface of the metal is covered with sand, slag, or other badly conducting material; the feeding-hole is closed, and the gas supply-pipe opened. The pressure exerted depends upon the temperature to which the liquid acid is exposed. This is regulated by immersing the bottle in a cistern of water, which can be heated when necessary.

Of the methods for producing homogeneity in the molten metal by agitation, that of Mr. Allen is the most noticeable. This consists of an apparatus resembling a ship's screw-propeller, covered with refractory material, which is attached to a strap capable of being revolved by appropriate gearing, which is lowered into the metal in the ladle, and turned rapidly for a few minutes before casting. This was introduced at Sheffield in 1881, but has not been much used elsewhere. Another plan introduced by Mr. James Riley in 1884, that of transferring the metal from the receiving-ladle to a second or casting-ladle, has been found to be extremely beneficial in promoting uniformity of composition in large charges.

A method of combining the Bessemer and open-hearth processes practised at Neuberg in Styria may be here noticed. The converter-charge of manganiferous grey pig-metal, after blowing for nineteen minutes, when the carbon is reduced 0·10 per cent., is poured into a ladle and transferred to an open-hearth melting-furnace, previously well heated, where it lies for three or four hours, being kept on the boil during the whole time. Two or three additions are made of steel and iron scrap, to the extent of about 5 per cent. in all, and, finally, about 5 or 6 per cent. of ferro-manganese is added before casting. The steel so treated is said to be of equal value with crucible steel for almost all purposes. Steel is now very largely used for making wheels and other parts of locomotives, which are cast to pattern in the same way as those of cast-iron; and even such large irregularly shaped objects as the stems and stern-frames of steamers are similarly produced. In these cases the castings are rendered sound by the addition of ferro-silicon or silico-ferro-manganese, a process known as steel-casting without blow-holes, which was originally introduced at Terre-Noire in France. Special precautions are also required in moulding; the surface of the mould must be very refractory and highly polished, while possessing a certain permeability to gases. At Gratz in Styria, rails and castings of parts of locomotives are sometimes made from the same charge of open-hearth steel. In such cases the rail-ingots are poured first, and the metal remaining in the ladle is tempered with ferro-silicon for the pattern castings. The amount of silicon necessary for the latter purpose is 35 lbs. per 5½-ton charge, which is added in the form of 350 lbs. of 10 per cent. ferro-silicon, previously melted in a crucible and poured into the ladle. The average composition of the casting is—

Carbon, 0·4–0·5; silicon, 0·3–0·4; manganese, 0·45–0·6; phosphorus, 0·06–0·09.

The tensile strength is about 38 tons per square inch, with 10 to 14 per cent. elongation, and 20 to 40 per cent. contraction on fracture; such castings must be annealed before they can be used.

The action of silicon in consolidating steel castings is not well understood. Pourcel considers it due to the reduction of oxide of iron in the metal with the production of silica, which combines with iron or manganese oxides to form a slag without giving off any gas; while, on the other hand, when a carburized tempering material, such as spiegeleisen, is used, the reduction of the oxide of iron being effected by carbon, carbonic oxide is evolved, and may be entangled in the casting, rendering it unsound. Müller considers the effect to be mainly physical; the silicon, by reducing the solvent power of the metal for hydrogen, promotes the separation of that gas while it is still liquid.

*Manipulation of Steel Ingots.*—When the ingots have solidified in the casting-pit, they are stripped by removing the moulds by the ingot-crane, which in most cases is readily done, as the ingots are broader below than above. When, however, the mould cannot be detached, it is lifted with the enclosed ingot and placed upon the framing of a hydraulic press or hammer adjoining the pit, where the "sticker" is forced out by the ram acting from above. Formerly ingots were invariably hammered before being passed through the rolling-mill, but the practice is now generally given up, except for the very heaviest forgings, and even with these the hydraulic forging-press (see p. 309) is coming gradually into use, and will probably supersede the hammer. For working the ingots, they must be brought to a full orange-red heat, which is usually done in a re-heating furnace of the Siemens, Bicheroux, or some analogous form. These are made of considerable length, with the head sloping from the flue end towards the fire-bridge at about 1 in 9 or 10, so that the ingot introduced at the upper or colder end may be gradually rolled downwards until raised to the required heat, which requires about three hours. If intended for rails, it is then passed through the cogging-mill and reduced to a parallel-sided bloom, which when re-heated is rolled to the proper section at one operation in the rail-mill. The rails are sawn to the proper length, giving a short piece or crop from either end. These are in some cases sold for wire-drawing or other uses, but more generally they are re-melted in the open-hearth furnace; in any case, they are a source of waste and loss in the manufacture. It is, therefore, customary in rail-making to diminish the proportion of crop-ends by using heavy ingots, which allow several lengths of rail to be cut from a single rolled bar. Thus at Eston four 30-feet rails of the heaviest section are made from a single ingot of 35 cwts. with only one pair of crop-ends. At Workington a three-rail ingot is passed nine times through the cogging-mill and thirteen times through the rail-mill to produce the finished section.

*Gjers' Soaking-Pit.*—An important economy has been realized in the rolling of steel ingots by utilizing the heat given out by the metal on solidification for the purpose of bringing it to the proper temperature for cogging and rolling without the use of the re-heating furnace. This is done by the so-called soaking-pit, introduced in 1882 by Mr. John Gjers, which consists of a square pit or cell, a little wider than the base of the ingot and somewhat deeper than its length, having an inner lining of fire-brick 9 inches thick. The ingot when stripped from the mould is removed by the crane as soon as it can be handled with safety, but before the metal has completely solidified throughout, and placed in the pit and closely covered. The heat given out by the ingot, which under ordinary circumstances is dissipated in the atmosphere, is absorbed and stored up by the brickwork; and supposing the latter to be cold, the heat of two ingots will be sufficient to warm it up sufficiently to bring subsequent ingots to a uniform heat for rolling in about an hour. When the brickwork becomes overheated, it may be cooled by dropping in a lump of bituminous coal, when the surplus heat is absorbed in the gasification of the volatile contents.

The soaking-pits are usually arranged in groups of six in any convenient position between the casting-pit and the rolling-mill. Their application is illustrated in figs. 115, 116, which represent the arrangements adopted at the Blochairn Works of the Steel Company of Scotland, where open-hearth steel is made on a very large scale for ship-building purposes. The melting-furnaces, twelve in number, are arranged in a row, having a line of railway parallel to them, upon which a locomotive draws a carriage, *a*, fig. 116, conveying a ladle for the steel and a mould for the slag. About the middle of the row of furnaces is a 20-ton hydraulic lift, *b*, which raises the ladle until a pair of horns on each side of the pouring-lip come in contact with brackets, which are so placed that it is kept in the same horizontal plane until the whole contents are poured into the second or casting-ladle, *c*, which is attached to a central crane like that of a Bessemer pit, but which has no lifting motion. The charge when transferred is brought over the moulds in the casting-pit, *d*, and cast in the usual way. This method of transfer is found to be beneficial in mixing the contents of the charge, and produces great uniformity in the ingots, while the time of casting is about fifteen minutes less than by the old method. The pit is 40 feet in diameter and 3 feet deep. It has three ingot-cranes, *e*, with lifting powers of 2 and 3 tons, which may be used separately or combined. Between the cranes are two sets of soaking-pits, *f*, each with six cells, varying in capacity for ingots from 30 cwts. to 3 tons each. Two of the three cranes lift the ingots from the moulds and deposit them in the soaking-pit, while the third or central crane lifts them when soaked or re-heated and places them on a tipping-carriage, *g*, at the end of the feed-rollers of the cogging-mill. This carriage is worked by a hydraulic engine, so that the ingot is deposited without shock on the feed-rollers of the mill.

The cogging-mill, *h*, has hollow steel rolls, with 32-inch centres and edging-grooves on each side. The top roll is balanced by hydraulic presses placed under the bed-plate, and has a lift of 14 inches. The setting screws for the rolls are worked by screw-and-worm wheels driven by a Willans steam-engine. The ingot is turned from flat to edge, or *vice versâ*, when on the feed-rolls, by arms put in motion by hydraulic engines placed below them.

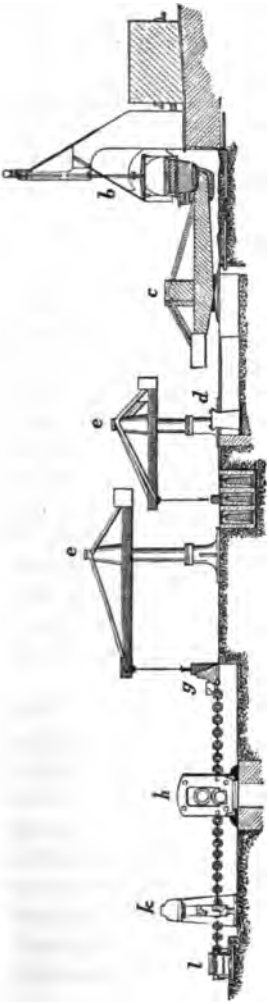

Fig. 115.—Casting-Pit and Cogging-Mill; longitudinal elevation.

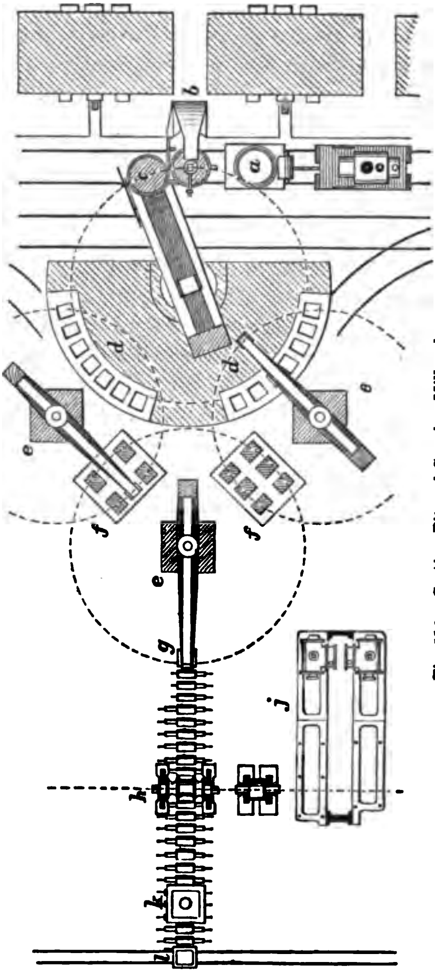

Fig. 116.—Casting-Pit and Cogging-Mill; plan.

The mill is managed by one man and two boys, one of whom attends to the setting and the other to the turning- and feeding-gear. The mill-engine, *j*, has two 36-inch cylinders, 4 feet stroke, and is geared in the proportion of 2¾ to 1. The rolls make 28 revolutions or 180 feet circumferential speed per minute. The ingots remain about twenty minutes in the mould after casting, when they are stripped and transferred to the soaking-pit, where they remain about 55 minutes, being then passed through the cogging-mill and reduced to section, which operation, together with the subsequent shearing to the specified weight, requires about five minutes. The hydraulic guillotine shearing-machine, *k*, having a working pressure of 1,000 tons, and capable of shearing slabs 2 feet broad and 8 inches thick, is placed behind the mill, and has feed-rollers, which pass the sheared slab on to the bogie, *l*, whence it is carried, on rails, to the re-heating furnace of the plate-mill.

*Manufacture of Compound Plates.*—Armour-plates having a steel face upon a wrought-iron backing are made by running molten steel upon malleable iron plates. For this purpose a face-plate of steel containing about 0·5 per cent. carbon is attached by steel screws to soft iron plates, so as to leave a space between them to receive the liquid steel. The mould so prepared is heated to redness, and placed on edge in a moulding-box sunk in a casting-pit, the wrought-iron backing being in contact with one side of the box, while on the steel side a layer of sand is interposed; the spaces at the ends are also rammed with sand. When the mould is ready, open-hearth or Bessemer steel is run, as hot as possible, from the ladle into a trough with several feeding-holes discharging into the space between the plates. Owing to the very high temperature of the liquid steel, the surface of the iron plate is sufficiently softened to effect complete union between the two metals. The compound slabs, when sufficiently cooled, are removed to the rolling-mill and finished to the proper size, the screws and pieces of steel forming the sides being removed in the final shearing. The proportions adopted for the slab of a compound plate 8 inches thick when finished, are 12 inches for the wrought-iron foundation, 2 inches for the steel face-plate, and a space of 5 or 6 inches for the liquid steel; about one-third steel to two-thirds iron.

PARTIAL DECARBURIZATION OF CAST-IRON BY CEMENTATION.—The fact that articles of cast-iron become softened if imbedded in ferric oxide, and maintained for a considerable time at a high temperature, was published by Réaumur so long ago as 1722. The invention of this process is, however, generally ascribed to Mr. Samuel Lucas, to whom a patent was granted in 1804 for a method of softening cast-iron by cementation with "ironstone ore, or some of the metallic oxides, lime, or any combination of these." The castings to be softened are packed in cast-iron crucibles, containing finely powdered red hæmatite, and arranged in rows one above another in a furnace somewhat resembling the ordinary cementation chamber. When the furnace has been charged, all the openings are carefully closed and the fire is lighted, the temperature being gradually raised so as to reach a red heat in about twenty-four hours; the firing is subsequently continued during from three to five days, according to the thickness of the layer of malleable metal required.

When withdrawn from the furnace, articles which have been subjected to this treatment present the appearance of ordinary malleable iron, but are lighter in colour; their fractured surfaces are white and finely granular, and occasionally present a silky appearance not unlike that exhibited by soft steel. When the thickness of the object is at all considerable, a kernel of unchanged cast-iron is frequently left in the centre; this may sometimes be broken by bending without occasioning the rupture of the external skin of malleable iron.

It is essential for the success of this process that the metal used for the castings should be as free as possible from silicon, and not contain much manganese.

The principal application of this process is to small articles of hardware, such as keys, buckles, gun-furniture, stirrups, bits, &c. The stratum of malleable metal thus obtained on the surface of cast-iron may be externally converted into steel by a process of case-hardening, so that the same object may, at different depths, be successively composed of cast-iron, wrought-iron, and steel; cheap articles of cutlery, prepared in this way, are known in the trade as *run-steel* goods.

In the modification known as the Siemens ore-process, as distinguished from the scrap or Martin process, the bath of pig-iron is decarburized by the addition of hæmatite in lumps, which causes a violent boiling. When the metal is nearly of the right temper, it is allowed to rest, so that the slag may separate, and a little limestone is added to recover some of the iron. From 20 to 24 cwts. of ore are used in a 5-ton charge, about one-half of it being reduced and recovered in the ingots. About 1 per cent. more spiegel and 1 cwt. per ton more coal are required than in the scrap process. Usually the two methods are combined, the ore being mainly used for tempering the steel if too hard.

A process intended for the production of small articles that have been hitherto made as malleable castings has been recently introduced from Sweden by Mr. T. Nordenfeldt. The material, called *mitis-metal*, which is essentially wrought-iron, is melted in crucibles, the very high temperature required being obtained by the combustion of petroleum-vapour in specially constructed furnaces, and the requisite fluidity is obtained by the addition of a small quantity of aluminium before pouring. The appearance of large objects made by this process when fractured, is very similar to that of steel castings made with an addition of silicon.

HARDENING AND TEMPERING STEEL.—All varieties of wrought-iron containing above 0·25 per cent. of carbon possess the property of becoming hardened by sudden cooling from a high temperature. Steel thus treated is found to possess a lower specific gravity than before hardening, but on being again heated, and allowed to cool gradually, its original density, softness, and malleability are restored.

In manufacturing objects of steel, the metal is filed or turned into the required form when in a soft state, and is subsequently hardened by being strongly heated and rapidly cooled.

In doing this, however, it is difficult to arrive directly at the exact degree of hardness best fitted for the purpose to which the instrument is to be applied, and it is therefore customary to give to the metal in the first instance a considerable degree of hardness, and afterwards to soften it by an operation called "tempering." In this the workman is guided by the various colours assumed by the surface of the metal during the progress of the operation, and when the proper colour makes its appearance, the object is suddenly cooled. These tints, some of which are extremely brilliant, are probably occasioned by films of oxide corresponding with considerable exactitude to the degree of heat to which the metal is exposed, and they consequently serve as a tolerably accurate guide in determining the hardness which the object will acquire on being cooled.

The following colours appear in succession on the surface of a plate of steel when exposed to a progressive heat. A piece of polished and hardened steel, subsequently heated to 220° C., has a faint yellow colour, and is well suited for lancets and other instruments requiring an extremely fine edge. When tempered at 230°, a faint straw-colour tint is obtained, which is well adapted for razors and surgeons' amputating knives. Steel seasoned at 243° is of a full yellow colour; this is tougher than the above, and is the tint to which penknives are usually tempered. At 255° it acquires a brownish-yellow tint, which is the colour best fitted for chisels and shears for cutting metals.

Axes and plane-irons are tempered at about 265°, which develops a brown shade intermixed with purple spots. For table-knives and cloth-shears a temperature of 277° is employed, which gives a purple colour to the metal so treated. For swords and watch-springs the metal is cooled when of a bright-blue colour; this tint very nearly corresponds with a temperature of 288°.

At 293° steel assumes a fine blue colour, and is at this stage well adapted for small shears and ordinary chisels; at 316° it takes a dark-blue colour, which is that best fitted for large saws, the teeth of which require to be bent by hammering.

The absolute strength and toughness of steel is much increased when the tempering is effected with oil instead of water. The large tubes used for ordnance are now generally tempered in this way. Pointed steel projectiles for piercing armour plates are cast in metal moulds, and hardened at the points by heating to redness and dipping in water until they are no longer visibly red, the final cooling being effected in an oil-bath containing at least four times as much oil by weight as the object to be immersed.

*Damascening*, by which a surface is obtained covered by a variety of figures resembling the water-lines on certain kinds of silk, is produced by repeatedly drawing out, doubling up, and welding together a bar composed of a mixture of steel and iron, and subsequently treating with an acid. When an article, such as a sword-blade or gun-barrel, made of this mixture is washed with a weak acid, its surface becomes in a greater or less degree unequally attacked, as the surface of the iron retains its metallic lustre, while that of the steel is left covered with a black firmly adherent coating of carbon. This gives rise to the peculiar wavy figures which may be observed on the once celebrated sword-blades of Damascus.

The production of the three principal staples of iron manufacture, namely, pig-iron, wrought-iron (chiefly as puddled bars), and steel ingots, in the more important iron-making countries of the world, for the years 1884 and 1885, was as follows:[1]—

Production of Pig-Iron, Wrought-Iron, and Steel.

| Countries. | Year. | Pig-Iron. | Wrought-Iron. | Steel. |
|---|---|---|---|---|
| | | Statute Tons. | Statute Tons. | Statute Tons. |
| United Kingdom . . | 1884 | 7,811,727 | 2,237,535 | 1,774,926 |
| | 1885 | 7,415,469 | 1,911,125 | 1,888,045 |
| United States . . . | 1884 | 4,097,869 | 1,724,795 | 1,550,880 |
| | 1885 | 4,044,526 | 1,597,955 | 1,712,274 |
| | | Metric Tons. | Metric Tons. | Metric Tons. |
| Germany . . . . | 1884 | 3,572,155 | 1,593,000 | 1,138,500 |
| | 1885 | 3,751,775 | 1,403,000 | 1,140,500 |
| France . . . . | 1884 | 1,855,247 | 1,449,064 | 509,516 |
| | 1885 | 1,628,941 | 1,432,500 | 527,048 |
| Belgium . . . . | 1884 | 750,812 | 471,040 | 185,916 |
| | 1885 | 714,677 | 454,227 | 146,189 |
| Austria-Hungary . . | 1884 | 796,620 | 346,534 | 225,752 |
| | 1885 | 760,000 | 355,412 | 217,690 |
| Sweden . . . . | 1884 | 430,534 | 292,198 | 74,241 |

## Analysis of Cast-Iron, Wrought-Iron, and Steel.

Preparatory to its examination, the metal must be reduced to a suitable state of division, either by boring, turning, or planing; in the case of white-iron, it may be reduced to a coarse powder in a steel crushing-mortar. It is generally considered preferable, in order to obtain an average sample of a pig, to bore completely through it, so that a fair proportion of the graphite, which is occasionally found concentrated towards the centre, may be included in the borings. The borings obtained in this way are, when necessary, further reduced and thoroughly mixed by trituration in a Wedgwood or large agate mortar. In the analysis of pig-iron the proportions of the following constituents are usually determined, namely, carbon, distinguishing the graphitic from that in the combined state; silicon, sulphur, phosphorus and manganese; and in certain cases such metals as arsenic, lead, and copper are estimated.

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[1] In many cases the figures are approximations only, but are the best that can be obtained.

The amount of aluminium, magnesium, and calcium is sometimes also determined.

Sulphur.—About 10 grammes of the borings are slowly dissolved in concentrated hydrochloric acid, and the evolved gases may be passed through a solution of lead acetate slightly acidified with acetic acid; the sulphuretted hydrogen disengaged precipitates lead as sulphide, which is collected on a filter and washed, and subsequently converted into sulphate of lead, from the weight of which the percentage of sulphur is calculated. Instead of conducting the gases through a solution of lead acetate, they may be passed through an ammoniacal solution of nitrate of silver; silver sulphide will be precipitated, together with a small quantity of a dark-coloured compound resulting from the action of the evolved hydrocarbons on the silver salt. The precipitate is separated by filtration and dissolved in fuming nitric acid. Hydrochloric acid is added, and the whole evaporated nearly to dryness. The silver chloride is removed by filtration, and the sulphuric acid in the filtrate thrown down by solution of barium chloride.

The contents of the flask, after the metal has been fully acted upon, are transferred to a procelain basin and evaporated to dryness, the mass digested with concentrated hydrochloric acid, and water afterwards added. The insoluble residue, consisting of silica and graphite, is washed by decantation and collected in a platinum dish; the decanted liquid is reserved for the estimation of manganese, &c.

Carbon as Graphite.—The mixed silica and graphite are separated by the action of a warm solution of pure potash; the silica is dissolved, and the graphite (which remains insoluble) is washed with water, and dried by exposure for some time to a temperature of about 120° C., after which it is weighed. Upon subsequently burning the graphite in a muffle, it usually leaves a small quantity of a reddish ash, which must be deducted from the former weight; this, after fusion with nitre and sodium carbonate, may be separately examined.

Silicon.—The silica dissolved by potash is recovered in the usual manner by evaporation with hydrochloric acid; the residue is digested with acidulated water, collected on a filter, washed, dried, ignited and weighed. The amount of silicon in the iron is calculated from the silica obtained. After weighing, the silica may be examined for titanic oxide, which may also be present in the filtrate.

Manganese.—The hydrochloric acid solution, separated from silica and graphite, may be divided into two equal portions, one of which, representing five grammes of iron, is sufficient for the estimation of manganese. The iron in the liquid having been peroxidized by nitric acid, the solution must be neutralized by addition of sodium carbonate; sodium acetate is added, and the liquid boiled, when the iron will be completely separated as insoluble basic acetate. The filtrate containing manganese is rendered alkaline by ammonia, and, after the addition of a few drops of bromine, is boiled during from ten to fifteen minutes.

The hydrated oxide of manganese, which is thus separated from the liquid, is collected, washed, dried, ignited, and weighed as $Mn_3O_4$, which furnishes, by calculation, the quantity of manganese present.

The following rapid indirect method of determining manganese is used at Gratz. The nitric acid solution that has served for the Eggertz carbon determination is boiled with an addition of peroxide of lead ($PbO_2$) and filtered through asbestos, which gives a clear rose-coloured solution from the formation of permanganic acid, the amount of which may be determined by comparing the tint with that of a permanganate solution of known composition, or preferably by a standardized ferrous salt. Red-lead, a mixture of protoxide and peroxide of lead, is generally used instead of pure peroxide; but as the latter is the active agent, a larger quantity is necessary, and more nitrate of lead is produced.

PHOSPHORUS.—For the estimation of phosphorus about 5 grammes of the borings may be acted upon with warm nitro-hydrochloric acid in a flask with a long neck, and, after complete solution of the metal, the contents of the flask are transferred to a porcelain basin, and evaporated to dryness; the residue is moistened with concentrated hydrochloric acid and again heated to expel nitric acid. The residue is dissolved in hydrochloric acid, the solution diluted, filtered, nearly neutralized with carbonate of ammonium, and the iron in solution reduced to protoxide by the addition of sodium sulphite to the gently heated liquid; the subsequent addition of dilute sulphuric acid expels excess of sulphurous anhydride. Sodium acetate and a few drops of solution of ferric chloride are then added, and the liquid boiled; the phosphoric acid is thus precipitated as basic ferric phosphate with some basic acetate. The liquid is rapidly filtered with as little exposure to the air as possible, the precipitate slightly washed, and dissolved in hydrochloric acid, the solution neutralized with ammonium carbonate, and a mixture of ammonia and ammonium sulphide added; it is then gently heated to insure the conversion of the phosphate of iron intb sulphide. The latter is afterwards removed by filtration, washed with dilute sulphide of ammonium, and the phosphoric acid precipitated from the solution in the usual manner as ammonio-magnesium phosphate, and weighed as magnesium pyrophosphate, from the weight of which the amount of phosphorus is calculated.

When the amount of phosphorus present is small, the following process may be advantageously employed for its determination. Dissolve about 5 grammes of the metal to be examined in nitro-hydrochloric acid, and evaporate nearly to dryness; take up again by addition of a few drops of nitric acid, dilute with water, filter, add molybdate of ammonium, and allow it to stand twenty-four hours in a warm place. The yellow precipitate which falls is separated by filtration, washed with a weak solution of ammonium nitrate, and dissolved in dilute ammonia slightly warmed; the phosphoric acid is re-precipitated by addition of the usual magnesium solution, and the resulting salt ignited and weighed as magnesium pyrophosphate.

COMBINED CARBON.—In order to determine the amount of combined carbon, 5 grammes of the metal may be dissolved in an acid solution of cupric chloride, or preferably in a neutral solution of a double chloride of copper and sodium, or of copper and ammonium; the insoluble residue which remains after the complete action of this solvent, is collected and washed, and, when dried, submitted to combustion with cupric oxide in a current of oxygen; a gas combustion-furnace is most conveniently employed for this purpose. The total amount of carbon in the metal is calculated from the weight of carbonic acid (carbonic anhydride) absorbed by solution of potash in the usual manner. The carbon existing in a state of combination will be represented by the excess afforded by this process over that of the direct estimation of carbon in the form of graphite, as already described.

Instead of operating as above directed, about 5 grammes of the metal, in small pieces, may be introduced into a flask, covered with water, and iodine added. The mixture is, from time to time, shaken, and is allowed to stand until all the free iodine has been taken up; more iodine is now added, care being taken to prevent heating; for every part of iron operated on about five parts of iodine are required. If the metal has been reduced to a finely divided state, the operation will be complete when the whole has been dissolved. When, on the contrary, the iron has been used in the form of chippings of considerable size, the action may be arrested as soon as a sufficient weight has entered into solution. In this case the portions remaining undissolved must be taken out, carefully washed, dried, and re-weighed; the second weighing, deducted from the first, will represent the weight of the dissolved metal.

The solid residue is separated from the brown solution by decantation, and is then carefully washed by the same means; the final washings are filtered through a tube, of which the end is drawn out and closed by a plug of asbestos. After being finally washed it is dried, and when dry the tube with its contents, together with the remaining portion of dry solid residue, is introduced into a porcelain or hard glass tube. mixed with cupric oxide, and its combustion effected by a current of oxygen. The resulting $CO_2$, after passing through a tube containing calcium chloride, is absorbed by caustic potash and weighed in the usual way. From the weight of $CO_2$ obtained, the amount of total carbon is calculated. From this must be deducted the amount of graphitic carbon, previously determined, and the difference will represent the combined carbon present. Bromine may be employed in place of iodine, but the results obtained are not so accurate, being generally too low.

According to Sir F. Abel,[1] when steel is dissolved by digestion in chromic acid mixed with sulphuric acid without heating, a residue is obtained containing iron and carbon, whose composition is represented approximately by the formula $Fe_3C$ (Fe 93·3, C 6·77 per cent.) The proportion of this substance varies with the state of the metal, which, if soft or annealed, yields it in sufficient quantity to account practically for the whole of the contained carbon; while, on the other hand, none is obtained from hardened steel, the carbon being then entirely combined. With tempered steels the amount of carbide varies with the heat employed in tempering, thus at a straw tint 30 per cent., and at a deep blue 42 per cent. of the total carbon present were obtained in this condition. A similar substance was obtained by Müller from spiegeleisen, and described under the name of *amorphous iron*.

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[1] 'Proceedings of the Institute of Mechanical Engineers,' 1885, p. 46.

MINUTE TRACES OF FOREIGN METALS.—About 30 grammes of the iron or steel should be employed in the examination for metals precipitated by sulphuretted hydrogen, *e.g.*, lead, copper, &c. The metal to be examined is dissolved in hydrochloric acid, and the solution, diluted, partly neutralized with sodium carbonate, and submitted to the action of sulphuretted hydrogen. After saturation with the gas the liquid is allowed to stand at rest for several hours, and the small quantity of precipitate which subsides is examined for the various metals by ordinary analytical processes.

Chromium and vanadium are to be looked for in the carbonaceous residue obtained by dissolving a considerable quantity of the iron in weak acids; aluminium, calcium, and magnesium may be estimated in the filtrate by the usual processes.

## EGGERTZ'S PROCESSES.

*Determination of Carbon.*—This is a process founded on the use of standard solutions, and is based on the fact that when iron containing carbon is treated with nitric acid slightly diluted and warm, the combined carbon is converted into a dark brown colouring-matter, while the graphitic portion remains unattacked.

By addition of water the solution may be brought to the colour of a standard liquid, obtained by dissolving a given weight of a steel of known composition, and the proportion of carbon in the metal under examination is subsequently determined by measuring the volume of its solution.

Pure nitric acid diluted to a density of 1·20 is employed, and the quantity of metal operated on is usually 0·10 gramme. The steel, in the form of filings which have been previously passed through a metallic sieve, of which the meshes are less than 0·004 inch in diameter, is attacked, in a test tube or small flask, by acid of the density above specified. If the metal contains but little carbon, from 1·5 to 2 c.c. of acid will be sufficient for the solution of 0·10 gramme of filings, but if the amount of carbon be large, as in the case of spiegeleisen, from 4 to 5 c.c. will be required. By the aid of a moderate heat solution is almost immediately effected, attended by effervescence, and black flocks, in greater or less abundance, will be observed floating in the liquid.

In order to obtain uniform results, it is necessary that the trials should be conducted at the same temperature and under similar conditions. To this end the tube in which the solution is being prepared is placed in a water-bath and kept constantly at a temperature of about 80° C. The black flocks, above alluded to, are seen gradually to dissolve with evolution of bubbles of gas, and the liquid becomes proportionately darker in colour.

At the expiration of three hours complete solution is effected, and the tube and its contents are rapidly cooled by being plunged in cold water. The liquid is then poured into a burette, graduated to tenths of a c.c. Finally, it is diluted with water until its colour exactly corresponds with that of the standard solution obtained by dissolving an equal weight of steel of which the composition has been previously ascertained; this solution, for comparison, must be contained in a tube having the same diameter, and made of similar glass to that of the burette. The similarity of colour may be judged by comparing the two by transmitted light, holding them between the eye and a window, or, still better, by placing the tubes side by side before a sheet of white paper placed opposite the light. After a little experience, a degree of exactitude will be obtained, which is represented in volume by from one to two-tenths of a c.c.; this will indicate the proportion of carbon to within two-hundredths of 1 per cent., if the standard solution be prepared by making it up to as many c.c. as the type steel contains tenths of 1 per cent. of carbon. As, according to Tunner's scale, the proportion of carbon varies 0·25 per cent. in passing from one No. of hardness to another, this is found in practice a sufficiently accurate approximation.

In order to obviate the necessity of making a standard solution for every set of determinations, various coloured liquids have, at different times, been employed, with a view of establishing a permanent scale. Caramel or burnt sugar, which has, among other substances, been employed for this purpose, gives various shades of brown and yellow, but they are by no means stable; partially decomposed solution of indigo in sulphuric acid is said to retain its colour for a considerable time. Hetmann recommends the use of a solution containing a mixture of potassium dichromate, and nitrate of cobalt; but, in the majority of cases, direct comparison with a standard solution of a given weight of steel, of known composition, is to be preferred.

The following determinations of carbon in various kinds of Swedish iron and steel are by Eggertz :—

| | | Per cent. of Carbon. |
|---|---|---|
| Softest Swedish Bessemer iron | contains | 0·08 |
| Soft steel | ,, | 0·75 |
| Best quality of cast-steel | ,, | 1·40 to 1·50 |
| Natural forge-steel | ,, | 0·99 ,, 2·44 |
| Cement-steel | ,, | 0·50 ,, 1·90 |
| Cast-steel | ,, | 0·86 ,, 1·94 |
| Hardest-melting cast-steel | ,, | 1·80 |
| Malleable cast-iron | ,, | 0·88 ,, 1·52 |
| Draw-plate steel | ,, | 3·30 |

This process cannot be employed for the comparison of steels obtained from different materials, or by different methods of treatment. In confirmation of this, Gruner states that at Neuberg a variety of coke pig yielded by the Bessemer process a steel corresponding in physical properties to No. 3 of Tunner's scale, while, according to the carbon it contained, it was only No. 6. This was a good ordinary cast-steel, very hard and difficult to weld, and by analysis, as well as by Eggertz's method, afforded only 0·3 per cent. of carbon; by analysis, however, it was found to contain nearly 1 per cent. of silicon, which, to a certain extent, may replace carbon in steel as well as in cast-iron.

*Sulphur.* — The ordinary methods of determining the amount of sulphur contained in iron and steel not only necessitate a considerable expenditure of time, but they also require an amount of analytical skill not always at command in establishments of limited extent. Eggertz has, therefore, sought a more rapid process, by which, without any pretence to great accuracy, an approximate estimation may be made of the amount of sulphur present in pig-iron, wrought-iron, and steel. The basis of this process is the more or less darkened shade acquired by a silver plate exposed to the action of the sulphuretted hydrogen evolved from an attack of a given weight of the metal under examination.

One gramme of distilled water and 0·5 gramme of strong sulphuric acid are poured into a stoppered bottle about 0·025 m. in diameter and 0·15 m. in height; into this, in the state of a finely divided powder, is introduced 0·10 gramme of the metal to be examined, and a piece of thin silver plate is immediately hung in the upper part of the flask, by a fine platinum wire retained between the neck of the flask and its glass stopper. At ordinary temperatures the metal will be completely dissolved in about fifteen minutes, and the silver plate may then be removed for examination. After numerous experiments, Eggertz has arranged a scale of colours corresponding to the varying amounts of sulphur present, estimated as hundredths of 1 per cent. This method of estimating small quantities of sulphur may be conveniently used for the determination of quantities of less than 0·10 per cent., and is applicable to pig-irons of high quality, such as those produced in Sweden; it is not, however, to be recommended for iron obtained with mineral fuel from the ordinary ores of this country.

*Silicon.*—The accuracy of determinations of silicon in iron and steel is not unfrequently impaired by the presence of a notable amount of intermingled slag, which, being a mere mechanical impurity, has no relation whatever with the chemical composition of the metal. In order to obviate this difficulty Eggertz has devised a method of estimating silicon in the presence of slags, which is based upon the fact that when iron is acted on by bromine or iodine it dissolves, and the silicon which is liberated is transformed into a form of silica completely soluble in a boiling solution of sodium carbonate, while that in combination as slag, should any be present, is not thus acted upon.

About 3 grammes of metal in the form of finely divided filings, previously sifted through a sieve of the degree of fineness specified when describing the determination of carbon, are treated with five times their weight of iodine in 15 c.c. of water, contained in a beaker of six or seven times that capacity. The water used should have been previously boiled, for the purpose of freeing it from air, and, as the operation should be conducted at a low temperature, the beaker must be kept cool, either by ice or by a current of cold water. As soon as the complete solution of the iron has been effected, the liquid is diluted to three times its original volume by a further addition of cold water, and, after being well stirred, is allowed to settle. The lighter portion of the graphitic carbon remains in suspension, and is, with the greater bulk of the solution, poured upon a wetted filter in a small glass funnel. To the heavy insoluble residue, which is retained in the beaker, a few drops of hydrochloric acid are added, and the whole is well stirred with a glass rod. If this should be followed by a disengagement of gas, it is proof that the whole of the metal has not been dissolved, and after the addition of a little sodium carbonate, to neutralize free acid, more iodine is introduced and complete solution effected. The whole of the residue is now transferred to the filter, and washed with distilled water until the addition of potassium ferrocyanide ceases to indicate the presence of iron. The filtrate is evaporated to dryness with addition of hydrochloric acid, for the purpose of recovering any traces of silica it may contain, and the original insoluble residue, which may contain graphite, silica, and unattacked slag, is transferred, without drying, to a large platinum dish, where it is treated with a saturated solution of sodium carbonate. After being heated for about one hour in a water-bath, during which time it is occasionally stirred with a platinum spatula, the alkaline silicate is carefully decanted from the insoluble residue upon a filter. A fresh quantity of sodium carbonate solution is now added, and, after heating during another hour in the water-bath, the whole is thrown upon a filter and carefully washed. The alkaline solution is now evaporated to dryness with the addition of hydrochloric acid, and to the dried residue hydrochloric acid is first added, and afterwards water. After boiling, the silica is separated by filtration, dried, ignited, and weighed. To the weight thus obtained is added that of the silica resulting from the evaporation of the iron solutions, and from the total is calculated the percentage amount of silicon present in the metal; the insoluble residue may contain graphite, slag, and titanic oxide.

By this process Eggertz has found that the amount of silicon in good bar-iron may vary from 0·01 to 0·10 per cent.; Krupp's steel afforded 0·03 per cent., while ordinary cast-steel, of good quality, contains traces only. Iron from a charcoal hearth, destined for the manufacture of wire, contained 0·33 per cent. of slag, while in armour-plates it amounts to from 0·75 to 3·00, and in rails sometimes to as much as 5·00 per cent.

## COBALT.

Cobalt is a metal of a steel-grey colour, and is susceptible of receiving a high polish. Excepting traces which occur in meteoric iron, cobalt is not found in the metallic state. It is reduced from its oxides by ignition with charcoal more easily than iron, and than are some of the more difficultly fusible metals; but when thus obtained it always contains carbon.

Cobalt in its purest state is obtained by the ignition of its oxalate. If either oxalate of cobalt, or a mixture of oxide of cobalt with charcoal, be strongly heated in a wind-furnace, with a little powdered glass, free from lead and from other reducible metals, a button of fused metallic cobalt will be obtained. Cobalt may also be reduced from its oxides by hydrogen; but unless the heat applied be considerable, the reduced metal is pyrophoric, taking fire in contact with air, and giving rise to the production of $Co_3O_4$. The spongy cobalt, obtained by ignition of its oxalate, after being allowed to cool in a closed vessel, may be collected in the form of a solid button by exposure to a high temperature in a lime crucible.

Cobalt is attracted by the magnet, and is capable of receiving a slight magnetic power when rubbed with a magnet; according to Pouillet, this power is not destroyed by the strongest red heat. Its specific gravity is from 8·54 to 8·70. It is not altered by the action of air and water at ordinary temperatures, but when very strongly heated takes fire, and is converted into the three-quarter oxide. It decomposes aqueous vapour at a red heat, and is dissolved by hydracids. By dilute oxygen acids it is, by the aid of heat, slowly dissolved with evolution of hydrogen gas. This metal unites by fusion with antimony and arsenic, the combination being attended by incandescence; the resulting alloys are brittle, and have an iron-grey colour. Metallic cobalt has a greater tenacity than iron, but is not employed in the arts.

### COBALT ORES.

The principal ores of this metal are the following:—

*Smaltine*, $CoAs_2$.—Occurs in octahedra, cubes, and dodecahedra, more or less modified. Colour, tin-white, inclining to steel-grey; fracture, granular and uneven; specific gravity, 6·4 to 7·2. This ore essentially consists of cobalt and arsenic, and is found in veins associated with silver and copper. Occurs in Cornwall; in Bohemia; at Freiberg, and, more abundantly, at Schneeberg in Saxony.

*Cobalt Glance*, CoAsS.—Lustre, metallic; colour, silver-white, inclined to red; streak, greyish-black. Occurs at Tunaberg, Riddarhyttan, and Håkansbo, in Sweden, in large, well-defined crystals; also at Skutterud, in Norway. It is likewise met with at Querbach in Silesia, Siegen in Westphalia, and at Botallack in Cornwall. The most productive mines are those of Vena in Sweden, which were first opened in 1809. This ore of cobalt is also found in California, &c.

*Cobalt Bloom.*—Occurs in thin oblique crystals, having a well-defined cleavage and foliaceous structure. It is also found as an incrustation on other minerals, and in compact reniform masses. Its colour is a pinkish-purple, resembling that of peach-blossom. When scratched it affords a greenish streak. This mineral is generally associated with silver and lead, and with other ores of cobalt, and is abundantly found at Schneeberg in Saxony, Saalfeld in Thuringia, and Riechelsdorf in Hesse Cassel. It is also found in England, in the counties of Cornwall and Cumberland, but does not occur in this country in sufficient abundance to render its extraction of commercial importance. Its percentage composition, according to Bucholz, is 39 of oxide of cobalt, 37 arsenic anhydride, and 22 of water: formula $Co_3As_2O_8.8H_2O$. When heated it gives off arsenical fumes, and, fused with borax, affords a bead of a fine blue colour.

*Mispickel* sometimes contains from 5 to 10 per cent. of cobalt.

## ESTIMATION OF COBALT AND NICKEL.

The ores of cobalt and nickel are usually very complex in composition; they are, however, almost invariably found together, and consequently the methods employed for their estimation and separation from one another, as well as from the various metals with which they are generally associated, will be described as one series of operations.

The ore is finely crushed, and, according to its richness, from 4 to 7 grammes may be taken for analysis. If the mineral contains much sulphur or arsenic, it is first roasted in a porcelain crucible in a muffle, in order to dispel the greater portion of these substances. The residue is then well boiled with hydrochloric acid, to which a little nitric acid has been added, until the metallic oxides are completely dissolved. The solution is slightly diluted with water, nearly neutralized with ammonia, more water added, and then boiled with an excess of acetate of sodium, by which iron and aluminium are separated in the form of basic acetates; the precipitate thus obtained will also contain arsenic. It is best to redissolve this precipitate, after washing, in hydrochloric acid, and, after neutralizing part of the free acid with ammonia, to precipitate a second time by acetate of sodium and boiling, as a small quantity of cobalt and nickel is generally carried down in the first precipitate of basic acetates. The two acetate solutions are now mixed and carefully neutralized with ammonia. They contain all the cobalt and nickel, and possibly some manganese, zinc, copper, bismuth, and lead. On passing sulphuretted hydrogen through the solution, cobalt, nickel, zinc, copper, bismuth, and lead are thrown down as sulphides, leaving the manganese in solution along with any earthy oxides present in the ore. These sulphides are collected on a filter, washed, dried, and roasted, dissolved in hydrochloric acid, and the copper, bismuth, and lead removed by passing a stream of sulphuretted hydrogen through the acid solution. The filtrate is now evaporated nearly to dryness, the salts taken up with water, a little acetate of sodium and a few drops of ammonia added, and then freely acidified with acetic acid. On passing sulphuretted hydrogen through this acetic acid solution the zinc is thrown down by itself as zinc sulphide. The filtrate from the sulphide of zinc contains the whole of the cobalt and nickel, which are finally precipitated as sulphides by making it alkaline with ammonia and again passing sulphuretted hydrogen. The sulphides of cobalt and nickel are thrown on a filter, washed, dried, thoroughly roasted, and weighed. After roasting, the nickel exists as $NiO$, the cobalt as $Co_3O_4$. These oxides may be reduced at a full red heat by means of hydrogen gas, and the nickel and cobalt weighed as metals.

It now only remains to separate the cobalt from the nickel. The oxides or the metals are dissolved in hydrochloric acid and the excess of acid driven off by evaporation. The chlorides are dissolved in water and the solution poured into a flask, with the addition of an excess of freshly precipitated barium carbonate, together with a few drops of bromine. The flask is loosely corked and the fluid allowed to stand six or eight hours, with frequent agitation. The cobalt is precipitated as peroxide, while the nickel remains in solution. The precipitated cobaltic oxide and the excess of carbonate of barium are well washed and dissolved in hydrochloric acid, and, after separating the barium by sulphuric acid, the cobalt is precipitated by potash. After washing, drying, and igniting, it is either weighed as $Co_3O_4$, or reduced to the metallic state by hydrogen. The filtrate from the cobalt, containing the nickel, is of a pure green colour. After removing the baryta by sulphuric acid, the oxide of nickel is thrown down by potash and weighed. From the weight of the oxide of nickel thus obtained, the percentage of this metal present is calculated. The yield of cobalt is usually returned as $Co_3O_4$.

## PREPARATIONS OF COBALT.

Two compounds of cobalt are extensively employed in the arts, namely, oxide of cobalt and smalt.

*Oxide of Cobalt.*—In the preparation of cobalt oxide ($Co_3O_4$) on a large scale, *speiss*, resulting from the fusion of arsenical ores of cobalt, is first subjected to calcination. The roasted speiss is subsequently dissolved in strong hydrochloric acid, and iron, arsenic, &c., precipitated by the gradual addition of milk of lime. When the precipitate thus obtained has subsided, the clear supernatant liquors are drawn off into vats, in which sulphuretted hydrogen is passed through them as long as metallic sulphides are produced. As soon as these have completely settled, the clear liquid is again drawn off, and oxide of cobalt precipitated by the addition of bleaching-powder; the hydrate, thus obtained, is heated to redness. Oxide of cobalt, which is largely made in Birmingham by the nickel-refiners, is employed in the Potteries and by glass-makers, enamellers, and others, who use it, either alone or in conjunction with various fluxes, for imparting a blue colour to their wares.

*Smalt.*—The preparation of smalt, which is a double silicate of cobalt and potassium, was invented in Saxony about the year 1550, and is perhaps the only process in connection with the ores of cobalt which can be strictly regarded as a metallurgical operation. Smalt is applicable to all purposes for which a cheap durable blue is required as a surface-colour. A pigment of this kind is attackable only by agents capable of decomposing glass; and smalt is, consequently, more permanent than the majority of colours.

The ore destined for the manufacture of smalt is first roasted in a reverberatory furnace having in communication with it chambers for condensing the arsenical fumes which are evolved. After having been suitably roasted in this furnace, the ore is mixed with pure siliceous sand and potassium carbonate. *Zaffres* are ores of cobalt, which contain a sufficient amount of silica to form a blue glass with the addition of carbonate of potassium only. The fusion of the mixture is effected in large earthen pots arranged in a furnace similar to that employed in the manufacture of plate-glass. From the great fusibility of the ingredients, the whole will have become completely melted at the expiration of eight hours, and during this time the mass is often stirred until the glass appears homogeneous, and a speiss containing a little cobalt and a considerable amount of nickel, together with arsenic, iron, &c., has sunk to the bottom. The smalt is now ladled out from the pots with a large iron ladle, and thrown into a reservoir through which a current of water constantly flows; by this treatment it becomes split into minute fragments, and its subsequent pulverization consequently much facilitated. When the pots have been nearly emptied, each ladleful withdrawn will consist of a mixture of speiss and smalt. The former, being completely liquid, readily separates from the more viscous glass which adheres to the ladle, whilst the metalliferous speiss is run into cast-iron moulds. These, during the time they remain hot, give off dense arsenical vapours, and are therefore placed in niches in the brickwork of the furnace, so as to be in direct communication with the chimney.

The deeply coloured blue glass, after being removed from the vats into which it has been thrown, is ground with water to the state of an impalpable pulp between granite millstones.

The blue pulp thus obtained is passed, in suspension in water, through a series of wooden vats, in which the coarser particles are first deposited, and where the powder which gradually settles is classified in accordance with its order of deposition. From these vats the pasty smalt, after being allowed to drain, is removed to drying-kilns, and is finally sifted through fine metallic sieves to remove any accidental lumps, and packed for the market.

*Cobalt Blue*, or *Thénard's Blue*, is prepared by precipitating a solution of nitrate of cobalt by phosphate of potassium, and adding to the resulting gelatinous deposit from three to four times its volume of freshly deposited alumina, obtained by the addition of carbonate of sodium to a solution of common alum. This mixture, after being well dried and calcined, affords, when properly ground, a beautiful blue pigment.

*Printers' Blue* is the colour used for printing the ordinary blue patterns on china. It is mixed with oil, printed on paper, and transferred to the biscuit-ware; the colour is developed during the process of glazing. This colour is prepared by fritting silicate of cobalt with nitre, and adding a little basic sulphate.

*Rinman's Green* is a permanent green pigment prepared by precipitating a mixture of the sulphates of zinc and cobalt with carbonate of sodium, and igniting the precipitate after careful washing. It may be also made by mixing a solution of nitrate of cobalt with either nitrate or oxide of zinc, and subsequently evaporating and igniting.

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## NICKEL.

This metal is closely allied to iron and cobalt, being associated with them, not only in meteorites, but also in the majority of its ores. Nickel is a silver-white metal, ductile and malleable, and but slightly more fusible than iron, which, according to Deville, it surpasses in tenacity. Nickel containing small quantities of carbon is more fusible than the pure metal. Its specific gravity is 8·50, but this may be increased by forging to 8·67. Nickel, previously heated, burns in oxygen gas like iron; the pulverulent metal obtained by the reduction of oxide of nickel by hydrogen at a low red heat is pyrophoric. When oxide of nickel is strongly heated with charcoal in a wind-furnace it becomes reduced to the metallic state, and, by combining with a portion of the carbon present, gives rise to the formation of a fusible carbide, which collects in the form of a button at the bottom of the crucible in which the fusion has been conducted. When treated either with hydrochloric or weak sulphuric acid, this metal dissolves with the evolution of hydrogen; it also dissolves readily in nitric or in nitro-hydrochloric acid.

Pure nickel is of a silvery white colour, and is both ductile and malleable; it is attracted by the magnet at ordinary temperatures, and is capable of becoming permanently magnetic, but loses these properties at 250° C. Nickel unites with other metals, forming alloys, of which the most important is that with copper and zinc, known as *German silver*. This is composed of a mixture varying from 13 to 31 per cent. of zinc, with from 40 to 66 per cent. of copper, and 12 to 31 per cent. of nickel, according to the purposes for which it may be required. Nickel is also much employed for coating other metals with a brilliant silver-white covering. A bath much used for the electro-deposit of this metal is prepared by dissolving 4 parts of nickel sulphate in 4 parts of aqueous ammonia and 150 parts of water, holding in solution 50 parts of acid sodium sulphite. The deposition of metal is effected by a very feeble current.

## NICKEL ORES.

The ores of nickel, with but few exceptions, have a pale colour and metallic lustre. In some respects they resemble those of cobalt, but are readily distinguished from them by not communicating a blue colour to borax when fused before the blowpipe. Specimens of native nickel are said to have been obtained from the Erzgebirge, but it is not found in sufficient quantities to be of commercial value.

*Copper-Nickel; Kupfernickel.* Hexagonal.—This is a mineral of a pale copper-colour, affording a brownish-red streak. It occurs massive, and has a metallic lustre. It is extremely brittle, and has a specific gravity varying from 7·3 to 7·5. This ore is essentially composed of 44 parts of nickel and 56 of arsenic; formula, NiAs. When heated before the blowpipe it gives off alliaceous fumes, and subsequently fuses into a pale-green globule, which darkens on exposure to the oxidizing flame. Copper-nickel is generally found associated with the ores of copper, silver, and cobalt, and is principally obtained from the mines of Saxony; small quantities have, however, been raised in this country, particularly at Pengelly, Fowey Consols, and St. Austell Consols in Cornwall, and at the Bathgate silver mine in Scotland.

*Garnierite and Noumeite.*—The minerals bearing these names are indefinite hydrated silicates of magnesium containing oxide of nickel, which have of late years been found in large quantities in New Caledonia. In some cases noumeite contains above 32 per cent. nickel oxide.

Among the other ores of nickel may be mentioned the following:—

*Rammelsbergite.* Rhombic.—Formula NiAs and $Ni_2As_3$, an arsenical ore, found at Reichelsdorf in Hesse Cassel, and at Schneeberg in Saxony. It contains from 20 to 30 per cent. of nickel.

*Gersdorffite*, formula $Ni(SAs)_2$, another arsenical ore, containing sulphur, occurring both massive and in cubical crystals. This mineral, which is of a steel-grey colour, is found in Sweden, in the Harz, and at Schladming, Austria. It contains from 20 to 38 per cent. of nickel, and has a specific gravity of about 6·7.

*Antimonial Nickel*, formula NiSb, containing about 32 per cent. of nickel and no sulphur; a portion of the nickel is usually replaced by iron. It is a pale copper-coloured mineral from Andreasberg.

*Millerite* is a brass-yellow sulphide of nickel, occurring in delicate capillary forms. It is found in small quantities in Bohemia, Saxony, Cornwall, South Wales, &c.; contains 64 per cent. of nickel, and has a density of 5·3; formula NiS.

*Pentlandite*, a double sulphide of iron and nickel, of a bronze-yellow colour, containing from 18 to 21 per cent. of nickel, is obtained from Southern Norway. A somewhat similar mineral, containing from 10 to 12 per cent. of nickel, has been discovered in the neighbourhood of Inveraray in Scotland, and is also noticed by Mr. King as occurring at La

Motte, Missouri, United States. Another sulphide of nickel containing bismuth has been found in some of the German mines, which have also produced specimens of arsenate of nickel of a beautiful apple-green colour.

*Emerald Nickel*, or *Hydrated Carbonate of Nickel*, usually occurs as an incrustation on other minerals. It is nearly transparent, of a bright emerald-green colour, and has a vitreous lustre. Another ore of nickel, of a brown or nearly black colour, and containing variable quantities of sulphur, is found in connection with ores of cobalt at La Motte.

## METALLURGY OF NICKEL.

The nickel of commerce is chiefly obtained from garnierite, from copper-nickel, from pyrites containing nickel, and from speiss or matte obtained as a secondary product during the treatment of nickeliferous ores. Different processes are employed for the preparation of metallic nickel, but the details of the various operations in use in this country are kept secret by the manufacturers.

Berthier dissolves either roasted speiss or roasted Kupfernickel, together with the quantity of iron found by previous experiment to be necessary for the removal of arsenic, in boiling nitro-hydrochloric acid, containing an excess of nitric acid, and evaporates to dryness. The residue is treated with water, which leaves a large quantity of undissolved ferric arsenate, and carbonate of sodium is added to the filtrate, which is kept constantly stirred until the precipitate begins to exhibit a green tint; by this means the remainder of the ferric arsenate will be thrown down, together with a portion of the cupric oxide. Should the precipitate, which is white when first deposited, not eventually become brown, it is an indication that the amount of ferric oxide present is not sufficient to effect the complete removal of the arsenic acid; ferric chloride must consequently be added, and the ferric oxide precipitated by the cautious addition of sodium carbonate. The filtrate is treated with sulphuretted hydrogen, and the clear liquid, separated from sulphide of copper, &c., is boiled with excess of sodium carbonate. The precipitate, consisting of a mixture of the carbonates of cobalt and nickel, is, after being thoroughly washed, diffused in water, and a current of chlorine passed through it as long as this gas continues to be absorbed. After exposure to the air for the purpose of allowing the escape of any excess of chlorine, the liquor is filtered, and from the filtrate so obtained oxide of nickel, free from oxide of cobalt, may be precipitated by an alkali.

Cloez (Jahresb. 1857, p. 619) dissolves finely pulverized and perfectly roasted Kupfernickel in strong hydrochloric acid, and adds an excess of acid sulphite of sodium. The mixture is afterwards vigorously boiled until the whole of the arsenic acid has been reduced to arsenious acid; the excess of $SO_2$ is driven off; sulphuretted hydrogen is subsequently passed through the lukewarm solution in order to precipitate arsenic, copper, antimony, lead, bismuth, &c., and the mixture is allowed to stand for twelve hours. The whole is now thrown upon a filter, and the filtrate evaporated to expel excess of acid; water is finally added and iron and cobalt precipitated, after oxidation with chlorine, by addition of carbonate of barium or carbonate of calcium. The dissolved baryta, or lime, is removed by sulphuric acid, and separated by filtration; carbonate of sodium added to the filtrate yields a precipitate of pure carbonate of nickel, which is subsequently ignited and reduced. Solutions of speiss in nitro-hydrochloric acid may be treated in the same way after first expelling the nitric acid by boiling with excess of hydrochloric acid.

The principal nickel-works in this country are situated in the neighbourhood of Birmingham, the details of the various operations being, as before stated, as far as practicable, kept secret. The general routine practised in these establishments is, however, understood to be nearly as follows:—The ore, or speiss, or a mixture of the two, is first melted in a reverberatory furnace, with addition of lime and fluor-spar as flux; the slags thrown away, the resulting matte, or speiss, finely ground, and subsequently roasted until arsenious fumes cease to be evolved.

The roasted product is now treated with hot hydrochloric acid, in which it becomes almost completely dissolved, the solution is diluted with water, the whole of the iron peroxidized, and the iron and arsenic precipitated by neutralizing the liquor and subsequently boiling. Sulphuretted hydrogen is then passed through the clear liquors separated from the precipitate. The precipitate by sulphuretted hydrogen is separated and washed, and the solution treated with chloride of lime (bleaching-powder), to which a little caustic lime is added to neutralize the liberated acid. By this means oxide of cobalt is precipitated, and, after being washed and ignited, is ready for the market. The nickel is, after the addition of milk of lime, precipitated by boiling the liquor from which the cobalt has been previously thrown down.

The reduction of nickel oxide thus obtained is sometimes effected by a process of cementation. For this purpose a number of cylinders of refractory clay are fixed vertically in a furnace, so that the flame may play around them on all sides. These are open at top, and terminate at bottom in truncated cones passing below the fire-bars, through which the charge is removed. The dried oxide of nickel, either in lumps or in small cubes, intimately mixed with powdered charcoal, is introduced at the top of these cylinders, and a strong heat externally applied. The reduced metal retains the form of the lumps or cubes of oxide introduced, and is from time to time withdrawn through openings in the bottoms of the cylinders; a fresh charge being at the same time introduced at the top, so that the operation becomes, to a certain extent, continuous.

The hydrated nickel oxide produced in the wet way is sometimes mixed in a pasty mass with about 5 per cent. of flour and a little syrup. This mixture, which has the consistency of dough, is beaten into a frame, and subsequently cut into cubes of something less than an inch square; these are dried, and afterwards reduced to the metallic state in crucibles or in tubes in which they are heated to whiteness whilst surrounded by charcoal-dust.

According to Aubel, nickel can be fused before the tuyer of a Rachette furnace, and Montefiore states it may be melted in comparatively large quantities in the apparatus devised by Deville and Debray for the fusion of platinum.

## COPPER.

This metal appears to have been known in remote antiquity, and, alloyed with about one-tenth of its weight of tin, was anciently employed for making edge-tools and for other purposes. Copper has a red colour, is very malleable, ductile, and tenacious, and when warmed or rubbed exhales a characteristic odour.

The copper of commerce is not chemically pure, but contains traces of other metals, such as arsenic, tin, and silver. Pure copper may be precipitated by electrical agency from a solution of a pure salt of that metal, and the variety of copper known as *best-selected* is very nearly pure; *tough-ingot* and *tough-cake*, particularly the latter, may contain traces of arsenic, tin, sulphur, &c.

Chemically pure copper may also be obtained by reducing cupric oxide to the metallic state by passing over it a stream of hydrogen gas while heated in a hard glass tube. Under these circumstances the reduction takes place below a red heat, and the metal which remains in the tube is found in the state of a powder, readily assuming a metallic lustre when rubbed between hard surfaces.

The specific gravity of this metal varies slightly, in accordance with the nature of the treatment to which it has been subjected, hammered or rolled specimens having a greater density than ordinary fused copper which has not been thus compressed. The density of copper varies between 8·76 and 8·96, and when heated to whiteness it gives off metallic vapours, which impart a green colour to flame.

When copper at ordinary temperatures is exposed to the action of dry air, its surface is not oxidized; but if acted on by a damp atmosphere, it becomes covered with a green basic carbonate, known as "verdigris."

Water is decomposed by copper when heated to whiteness in the presence of steam; oxide of copper is formed, and hydrogen is set free. A concentrated solution of hydrochloric acid attacks copper, when in a state of fine division, with considerable facility; but when the metal is exposed to its action in more solid masses, its solution is attended with some difficulty.

The presence of the stronger acids does not determine the decomposition of water by this metal. When dissolved in concentrated sulphuric acid, sulphurous anhydride is evolved. Nitric acid, even when cold and diluted with water, dissolves copper with great facility, and gives rise to the rapid evolution of nitric oxide, which, coming in contact with the air, produces large quantities of the characteristic red fumes caused by the resulting compound.

The tenacity of copper is less than that of iron, but greater than that of gold or platinum.

Sheet-copper is extensively used for covering ships, and for a vast number of other purposes. Copper is also largely employed for making tubing, vacuum-pans, stills, &c., as well as for culinary vessels of different kinds; as an alloy with zinc in the form of brass or yellow-metal, and, combined with tin, as bronze, bell-metal, &c.

## COPPER ORES.

NATIVE COPPER; *Cuivre natif; Gediegen Kupfer.* Cubic.—This metal frequently occurs in a native state, and is probably sometimes the result of electro-chemical influences, by which sulphate of copper arising from the oxidation of its various sulphides is caused slowly to deposit the metal it contains.

Native copper is most frequently met with in irregularly shaped masses, occupying fissures in the rocks in which it is found; but it sometimes also occurs in a crystalline state, in which case the crystals are cubes, octahedra, or some immediately derived form. Native copper is both malleable and ductile; has a red colour, metallic lustre, and shining streak; possesses no traces of cleavage, and readily fuses before the blowpipe into a well-defined metallic globule, which, on cooling, becomes externally coated with a thin layer of oxide. In some localities specimens of this metal occur in a pure state, but it more frequently contains traces of other metals, particularly of iron and silver.

Native copper is met with in the mines of Cornwall, Brazil, and Siberia, but most abundantly in those of Keweenaw Point, Lake Superior, where masses exceeding 400 tons in weight have been extracted. Splendid crystallized specimens are also procured from Siberia and from the island of Naalsö, one of the Faröe Isles, where it accompanies fibrous mesotype in amygdaloidal trap.

The minerals of which copper forms an essential constituent are numerous and important, but we shall mainly confine our attention to such as are entitled to be ranked among copper ores.

CUPRITE; *Ruby Copper Ore; Cuivre oxydulé; Rothkupfererz.* Cubic. —This oxide is remarkable for its brilliant cochineal-red colour, most distinctly seen in transparent and translucent specimens.

This mineral frequently occurs in well-defined crystals of a ruby-red colour; its lustre is semi-metallic, streak shining and reddish-brown, fracture hackly or sometimes conchoidal, and its cleavage parallel to the faces of the octahedron. When crystals of this mineral are opaque, they are sometimes of an iron-grey tint on the surface, but their peculiar red colour becomes apparent when they are reduced to the state of fine powder. This mineral has a density of 5·99; its composition is as follows:—

| | |
|---|---|
| Cu | 88·80 |
| O | 11·20 |

These proportions are represented by the formula $Cu_2O$.

Octahedral copper oxide is found in many of the Cornish mines; particularly in those near Redruth, and at the Phœnix mines, near Liskeard. Isolated crystals, sometimes an inch in diameter, were formerly obtained at Chessy, in the neighbourhood of Lyons; and many splendid specimens have been brought from Siberia. This oxide is also found in extremely slender reticulated crystals; specimens of this variety are occasionally obtained from the mines of West Cornwall.

**Melaconite**; *Black Oxide of Copper; Cuivre oxydé noir; Kupferschwarz.* Cubic.—In many copper mines a black substance is found, which stains the fingers when handled, and is principally composed of cupric oxide, CuO, mixed with various earthy impurities. Analysis shows that this substance sometimes contains sulphur and arsenic, and often considerable quantities of the oxides of iron and manganese.

From this circumstance it appears that black oxide of copper, which in many localities is obtained in sufficient abundance to render its extraction an important consideration, is the result of the decomposition of other ores, such as copper pyrites, and that the sulphur and arsenic which it still retains are merely the result of incomplete decomposition.

This mineral is commonly found disseminated among other ores of copper, and sometimes occurs in shining botryoidal concretions or dull friable masses.

**Redruthite**; *Vitreous Copper; Cuivre sulfuré; Kupferglanz.* Rhombic.—Disulphide of copper is of an iron-grey colour, and is often iridescent; found in crystals, but more frequently in compact lamellar masses; pseudomorphic crystals of this mineral after galena, have occasionally been observed. The specimens obtained from the Cornish mines, and especially from Cook's Kitchen, frequently present themselves in thin six-sided prisms. This ore is friable, slightly sectile, and when scratched affords a shining lead-grey streak.

When pure it may be readily cut with a knife, and is fusible in the flame of a candle. Its density varies, according to texture, from 5·5 to 5·8, and its crystals are frequently twinned.

Disulphide of copper is almost always contaminated with a certain amount of sulphide of iron, by which its hardness and fusibility are considerably modified. An argentiferous variety from Mexico is known as stromeyerite.

The composition of a specimen of this mineral, from Tellemarken, Norway, analysed by Scheerer, was found to be as follows:—

| | |
|---|---|
| S | 20·36 |
| Cu | 79·12 |
| Fe | 0·28 |
| | 99·76 |

Its composition is expressed by the formula $Cu_2S$.

Although in this country magnificent crystals of vitreous copper are obtained from the Cornish mines, they are nevertheless almost exclusively confined to that county; the more compact and massive varieties occur in Siberia, Saxony, and the Banat.

COPPER PYRITES; *Chalcopyrite; Cuivre pyriteux; Kupferkies.* Tetragonal.—This mineral is distinguished by its strong metallic lustre and brass-yellow colour. It usually occurs in amorphous masses, with an irregular and slightly conchoidal fracture: it is also found in mammillated, stalactitic, and botryoidal forms, as well as in crystals. Its specific gravity varies from 4·1 to 4·3, and when strongly heated on charcoal before the blowpipe it readily fuses into a dull-black globule, which, from the presence of iron, becomes magnetic. When mixed with sodium carbonate, and similarly treated, it yields a button of metallic copper. If dissolved in nitric acid or aqua regia, it affords a solution which, on the addition of ammonia, assumes a fine blue colour.

The following analyses give the composition of specimens of this mineral from two different localities:—

| | From Cornwall. Analysed by R. Phillips. | From Sayn. Analysed by H. Rose. |
|---|---|---|
| S . . . . . . | 35·16 | 35·87 |
| Cu . . . . . | 30·00 | 34·40 |
| Fe . . . . . | 32·20 | 30·47 |
| Gangue . . . . | 2·64 | 0·27 |
| | 100·00 | 101·01 |

Its composition may consequently be represented by the formula $Cu_2S.Fe_2S_3$, or $CuS.FeS$.

This mineral is found in lodes or veins, which frequently occur either in granite or in clay-slate, although it is also met with in serpentine, gneiss, and other rocks. It is most commonly associated with iron pyrites, blende, and galena, together with carbonates and other ores of copper.

The principal localities in which this valuable ore is found are Cornwall and Devon, in England; in Saxony; at Goslar, in the Lower Harz; at Schemnitz and Kremnitz, in Hungary; at Fahlun, in Sweden; in the Ural Mountains, in Russia; also in China and Japan, and in Australia; formerly in considerable quantities at Chessy, in France.

The Cornish copper ores, once so extensively treated in the neighbourhood of Swansea, are chiefly composed of this mineral, and constitute the chief portion of the copper ores raised in the United Kingdom.

Enormous deposits of iron pyrites, through which copper pyrites and other ores of copper are more or less thickly disseminated, are extensively worked in Spain and Portugal. This mineral, besides supplying the sulphur required for the production of nearly all the sulphuric acid manufactured in Europe, yields very large quantities of copper by the process of wet extraction.

**ERUBESCITE**: *Cuivre panaché; Buntkupfererz.* Cubic.—This ore, which holds a somewhat important position among copper-producing minerals, has a reddish-brown colour, and metallic lustre; its surface is commonly iridescent with different shades of blue, purple, and red, from which circumstance it is called *cuivre panaché* by French mineralogists.

Fused before the blowpipe, it presents similar reactions to those obtained from copper pyrites, but when found in a crystalline form the crystals are either cubes or octahedra, of which the faces are not usually well defined. It occurs in the compact form, associated with other ores of copper, in Chili, Cornwall, Siberia, Silesia, Norway, and the Banat; also in the Cape Colony, and in the cupriferous shales of Mansfeld.

In this country the crystallized variety has, as yet, only been found in Cornwall, where, among other localities, it occurs in the neighbourhood of Redruth.

This, like copper pyrites, is a double sulphide of copper and iron; analyses of two specimens afforded the following results:—

| | From Cornwall; Varrentrapp. | From Killarney; Phillips. |
|---|---|---|
| Cu . . . . | 58·20 | 61·07 |
| S . . . . | 26·98 | 23·75 |
| Fe . . . . | 14·84 | 14·00 |
| Gangue . . . | ... | 0·50 |
| | 100·02 | 99·32 |

The specific gravity of the crystallized varieties varies from 4·9 to 5·1, and the faces of the crystals are, in many specimens, slightly curved; formula, $3Cu_2S.Fe_2S_3$ or $Cu_3FeS_3$.

**TETRAHEDRITE**: *Cuivre gris; Fahlerz.* Cubic; tetrahedral.—Usually occurs massive, but sometimes crystallized in well-defined tetrahedra. Its colour varies from steel-grey to iron-black, and when scratched it yields either an unchanged or a slightly brown streak. It has a conchoidal fracture, and sometimes an imperfectly developed cleavage parallel to the faces of the octahedron. It is brittle, and has a density varying from 4·6 to 5·1.

Dana believes that the general composition of this mineral may perhaps be represented by the formula $4Cu_2S + Sb_2S_3$ or $Cu_8Sb_2S_7$, in which each of the different metallic constituents may be, to a greater or less extent, replaced by the substitution of other isomorphous elements; so that sulphide of arsenic may be substituted for sulphide of antimony, sulphide of silver for sulphide of copper, &c.

This mineral frequently contains zinc and silver, and occasionally mercury. The following analyses of different specimens of this ore will serve to illustrate its very variable constitution:—

| Locality. | S. | Sb. | As. | Cu. | Fe. | Zn. | Ag. |
|---|---|---|---|---|---|---|---|
| From Clausthal; Rose . | 24·73 | 28·34 | ... | 34·48 | 2·27 | 5·55 | 4·97 |
| ,, Wolfach; Rose . | 23·52 | 26·63 | ... | 25·33 | 3·72 | 3·10 | 17·71 |
| ,, Corbières; Berthier | 25·30 | 25·00 | 1·50 | 34·30 | 1·70 | 6·30 | 0·70 |
| ,, Gersdorf; Rose . | 26·33 | 16·52 | 7·21 | 38·63 | 4·89 | 2·76 | 2·37 |
| Locality not named; Klaproth . . . | 10·00 | ... | 14·00 | 48·00 | 25·50 | ... | 0·50 |

Some of the finest crystals of this substance have been obtained from mines near St. Austell, in Cornwall; and very beautiful complex crystals of a bright polished aspect are found at Andreasberg, in the Harz; Kremnitz and Kapnik, in Hungary; Freiberg, in Saxony; and Dillenburg, in Nassau.

Blue Carbonate of Copper; *Azurite; Kupferlasur.* Monoclinic.—This mineral, which occurs both in mammillated concretions and in well-defined and brilliant crystals, is of a beautiful blue colour, and is sometimes perfectly transparent, although commonly translucent only. Its specific gravity varies from 3·5 to 3·7; lustre, vitreous or adamantine; fracture, conchoidal, and streak of a somewhat lighter blue than the mineral itself. When acted on alone before the blowpipe it is melted by the oxidizing flame into a black globule. By the reducing flame a bead of metallic copper is obtained. It dissolves with effervescence in nitric acid, and yields a solution affording all the common reactions of copper. When fused with borax in the oxidizing flame a glass of a bright-green colour is produced.

Its composition, according to analyses by Phillips and Karsten, is as follows:—

| | Specimen from Chessy; R. Phillips. | Specimen from the Banat; Karsten. |
|---|---|---|
| $CuO$ . . | 69·08 | 69·08 |
| $CO_2$ . . | 25·46 | 25·72 |
| $H_2O$ . . | 5·46 | 5·20 |
| | 100·00 | 100·00 |

The above numbers correspond to the formula $2(CuO.CO_2) + CuO.H_2O$, or $2CuCO_3.CuH_2O_2$. This mineral usually occurs associated with the red oxide and green carbonate of copper. Some of the chief localities from which blue carbonate of copper has been obtained are Chessy near Lyons, Siberia, and the Banat. Specimens of this ore are also found at Redruth, in Cornwall; Alston Moor, in Cumberland; in the Cuban mines, and in large quantities at Burra Burra, South Australia. When obtained in sufficient quantity, this substance constitutes a valuable ore of copper.

**Malachite**; *Cuivre carbonaté vert; Malachit.* Monoclinic.—Green carbonate of copper is remarkable for its fine emerald-green colour, of which the same specimen usually exhibits a great diversity of shades. When in a crystallized state, this substance is found in various forms derived from the oblique prism; but it is more frequently met with as mammillated, reniform, and amorphous deposits. It is likewise found in stalactiform masses, made up of several successive layers, of which the extent and thickness are apparent and well defined.

Malachite is found in considerable quantities in the Ural Mountains; in the mines of South Australia; formerly at Chessy in France; in the old mine at Sandlodge in Shetland; in the Banat; the Tyrol, and in some of the Cornish mines. It is, from its high percentage of metal, a valuable ore of copper, but it is also highly prized by the lapidary for various ornamental purposes.

The density of this mineral varies from 3·6 to 4·1; lustre, adamantine, inclining to vitreous; streak of a rather paler green than the mineral itself.

Its percentage composition is as follows :—

| | From Siberia; Vauquelin. | From Siberia; Klaproth. |
|---|---|---|
| CuO . . | 70·10 | 71·70 |
| $CO_2$ . . | 21·25 | 20·50 |
| $H_2O$ . . | 8·45 | 7·80 |
| | 99·80 | 100·00 |

The above numbers indicate that the composition of this mineral may be represented by the formula $CuO.CO_2 + CuO.H_2O$ or $CuCO_3,CuH_2O_2$. Malachite is advantageously employed for mixing with the various sulphides of copper during the operations of smelting. It is also sometimes used by artists as a green pigment; it affords a valuable material for the manufacture of the various salts of copper, and may be converted into blue vitriol by solution in dilute sulphuric acid and subsequent crystallization.

**Dioptase**; *Achirite; Kupfer-Smaragd.* Hexagonal.—Composition, $CuO.SiO_2 . H_2O$ or $CuSiO_4H_2$; colour between emerald-green and verdigris green. Occurs with quartz and calcite in limestone in the Kirghiz Steppe, and has been more recently found in the copper mines of Colorado and Arizona.

CHRYSOCOLLA; *Silicate of Copper; Cuivre Hydraté Silicifère; Kieselkupfer.* Cryptocrystalline; often resembling opal in texture; earthy. Incrusting various minerals or filling crevices, sometimes botryoidal. Accompanies other ores of copper; chiefly occurs near the surface. Colour, mountain-green, bluish-green, passing into sky-blue or turquoise-blue.

Found in the copper mines of Cornwall, Hungary, and Tyrol; in Saxony, Bavaria, Australia, on Lake Superior, and at various other localities in North America.

Its composition varies considerably, from the presence of impurities, as is generally the case with amorphous minerals, resulting from alteration.

Analyses of two specimens of this mineral from different localities afforded the following results:—

| | From Coquimbo, Chili; F. Field. | From Cornwall; Berthier. |
|---|---|---|
| $SiO_2$ . . | 28·21 | 26·00 |
| $CO_2$ . . | ... | 3·70 |
| CuO . . | 39·50 | 41·80 |
| $Fe_2O_3$ . . | 2·80 | 2·50 |
| $Al_2O_3$ . . | 4·97 | ... |
| $H_2O$ . . | 24·52 | 23·50 |
| Gangue . . | ... | 2·50 |
| | 100·00 | 100·00 |

Probable formula: $CuO.SiO_2.2H_2O$, or $CuSiO_3.2Aq$.

### DISTRIBUTION OF COPPER ORES.

Copper not only occurs in many different forms of combination, but its geographical distribution is very extensive, and its geological range equally wide. Ores of this metal are found in rocks of all ages, from the Laurentian to the Cretaceous, but their deposition would appear to have gone on with greater activity during the Permian period than at any other.

The principal portion of the copper produced in this country is obtained from the mines of Cornwall and Devonshire; but the Ecton mines, Staffordshire, at one time furnished considerable returns, and Parys mine in Anglesea once yielded large supplies. Wales has from time to time furnished a limited quantity of copper, and Ireland has contributed about 700 tons annually. The production of copper in the

United Kingdom has much decreased since 1862, when it amounted to 14,843 tons; in 1872 it had been reduced to 5,600 tons; and the present annual yield of the mines of this country probably does not exceed 3,000 tons.

In France there were formerly mines of considerable interest at Chessy near Lyons. These deposits, which occurred at the junction of mica-slate with Triassic and Jurassic rocks, largely consisted of azurite and cuprite; but after furnishing the cabinets of Europe with the finest known specimens of these minerals, they have become exhausted.

The most important copper-producing district of Prussia is that around Mansfeld, where mining has for centuries been carried on in the Kupferschiefer, immediately beneath the Zechstein. The copper-bearing stratum seldom exceeds 18 inches in thickness, but extends with wonderful regularity over an area of many square miles; the portion which is smelted constitutes a comparatively small proportion of the seam, and contains copper in the form of enclosed particles of various disseminated sulphides. The proportion of copper in the ore treated averages about 2½ per cent., while that of silver does not exceed $\frac{1}{70}$th of 1 per cent. For several years past the various Mansfeld establishments, which are worked with consummate skill, have treated about 500,000 tons of schist annually, and, in addition to above 11,000 tons of copper, have yielded fine silver to the annual amount of about 154,000 lbs. avoirdupois. A small quantity of copper is produced in the neighbourhood of Siegen and in Lower Silesia. The production of copper ore in the German Empire in 1881 reached a total value of £716,495.

The principal copper mines of the Russian Empire are in the Ural Mountains, the Altai, the Caucasus, and in Finland; but the latter are of minor importance. The copper ores of the Caucasus are said to be abundant, and there is evidence of their having been worked at a very early period; the present yield of these mines, as well as of those in the Altai, is inconsiderable. There are mines in the Ural Mountains, as well as on their western flank, where certain beds of Permian age are cupriferous, and possess a remarkable analogy with the Kupferschiefer of Mansfeld. The amount of copper furnished yearly by Russia is estimated at about 5,000 tons.

Upper Hungary and the Banat produce copper; the copper mines of the Schemnitz district have decreased in importance. The total production of copper in the Austrian empire including Hungary was in 1882 10,800 tons of ore, of a value of £63,600.

The quantity of copper furnished by the Scandinavian peninsula is small, but has somewhat increased within the last ten years. The mines of Alten, in Norway, are said to be in the most northern position of any in the world, being in latitude 70°. The mine of Vigsnæs is of considerable importance, as are also those of Røros, where the ore is disseminated in chloritic slate, forming metalliferous beds. The copper deposits of Sweden resemble those of Norway. There are eight groups of mines or mining districts, principally in the province of Dalecarlia. Fahlun has been celebrated for its copper mines; but its importance is much diminished, and it is now to some extent exhausted; the ores are poor, and do not yield above 4 per cent. of metal after being hand-picked. The annual production of copper in Sweden and Norway is estimated at about 3,000 tons.

The amount of copper now produced from Spanish pyrites is very large. The most remarkable deposits are those of Rio Tinto and Tharsis, both situated in the province of Huelva. These mines were extensively worked during the Roman occupation of the country, and subsequently by the Spaniards. Within the last twenty years, however, they have passed into the hands of powerful British companies, by whom they have been extensively and systematically developed.

In Portugal, at San Domingos, near the mouth of the River Guadiana, there are extensive mines of cupriferous pyrites worked by Messrs. Mason, Barry & Co., of London.

From these three mines some 550,000 tons of cupriferous pyrites are annually imported into this country, which on an average contain about 3 per cent. of copper. This pyrites is first burnt for the production of sulphuric acid, and the resulting *cinder* subsequently treated for copper by the wet process. The total annual production of copper from the Spanish and Portuguese mines, including copper-precipitate, was estimated in 1872 at 13,000 tons; at the present time their total yield is probably not very far short of 50,000 tons.

The only copper mines of any importance worked in Italy are those of Monte Catini, which are deposits for the most part enclosed in serpentine. A certain amount of copper ore of good quality was formerly exported from Turkey, and copper ores occur at Tenès and near Mouzaïa, in Algeria. At the latter place the veins are enclosed in rocks high in the geological series, belonging, it is believed, to the Cretaceous period.

Copper is found in the East Indies and in Japan; about 1,500 to 2,000 tons annually are said to be exported from the latter country. South Australia produces large quantities of rich copper ores, yielding metal of good quality; a large portion of this ore is now smelted in the colony. The celebrated Burra Burra mine, eighty-six miles from Adelaide, was first opened in 1845, and at once began to yield large quantities of the red oxide and green carbonate of copper. In 1850 the production from this mine was 18,962 tons of ore, averaging from 24 to 26 per cent. of copper. The production of the Burra Burra is now inconsiderable, but the total annual yield of the Australian colonies probably exceeds 12,000 tons of metallic copper.

Large quantities of copper ore have for some years been imported from the Cape of Good Hope. The value of the copper ores produced in Namaqualand during the year 1882 amounted to £331,546.

Among the important copper mines of Chili are those of Carrizal, north of the valley of Huasco, those of San Juan and La Higuera, between Huasco and Coquimbo, besides numerous others in the vicinity of Coquimbo. Large quantities of gold were obtained from the upper portions of the veins in this district previously to the close of the last century; as the production of gold fell off that of copper increased. The present annual production of copper in Chili and Bolivia is estimated at about 40,000 tons. A portion of the ore is smelted in the country, and the remainder either run into regulus, or exported in the raw state. The copper mines of Peru are but imperfectly developed, and the annual returns small. The mines of Cuba were formerly of great importance, but of late years their production has almost ceased. Copper ores are found scattered in considerable abundance throughout Mexico, but the mines of this metal are not worked to any considerable extent.

The most important copper-producing regions of the United States are those of Michigan on the southern shore of Lake Superior, and those of Arizona and Montana. On Lake Superior copper is found in trappean rocks and their associated conglomerates, which, for the most part, cover beds of sandstone, ascribed by Whitney to the Lower Silurian period. The most remarkable feature of this district is that the copper does not exist in the form of an ore, but almost exclusively as native metal. Masses of nearly pure copper weighing over 400 tons have sometimes been met with, and required to be cut with chisels into fragments of convenient size before they could be brought to the surface. The bulk of the produce is, however, obtained by stamping and washing rock containing from ¾ to 4 per cent. of copper. Copper pyrites occurs, to some extent, in the sandstones and limestones of the Mississippi Valley, but the deposits are not extensively worked. Copper-bearing veins are found in numerous localities, extending from Vermont to Tennessee, and are mined in various places. In Montgomery and Chester Counties, Pennsylvania, copper veins traversing New Red Sandstone and older metamorphic rocks have been sometimes worked. The total production of copper in the United States was, in 1872, 12,600 tons, since which date, by the opening of fresh mines, &c., chiefly in Arizona and Montana, the quantity has increased to nearly 75,000 tons.

In Canada there are the copper mines on the north shore of Lake Huron, and at Acton and Harvey Hill in the neighbourhood of Quebec; but the annual production of the Dominion is very small.

The present production of copper in the whole civilized world is estimated at but little short of 230,000 tons per annum, whilst in 1872 the quantity probably did not exceed 130,000 tons.

The table on p. 400, by Messrs. Henry R. Merton & Co., gives, with a very near approach to accuracy, the total production of copper during the years 1880 to 1885 inclusive. The figures marked with an asterisk are estimations only.

PRODUCTION OF COPPER.

| | | 1880. | | 1881. | | 1882. | | 1883. | | 1884. | | 1885. | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Tons. | | Tons. | | Tons. | | Tons. | | Tons. | | Tons. | |
| Algiers | | * 500 | | * 600 | | * 600 | | * 600 | | 260 | | 250 | |
| Argentine Republic | | * 300 | | 307 | | 800 | | 293 | | 159 | | 233 | |
| Australia | | 9,700 | | 10,000 | | * 8,950 | | * 12,000 | | 14,100 | | 11,400 | |
| Austria | | 470 | | 455 | | * 455 | | * 500 | | 670 | | * 670 | |
| Bolivia—Coro'coro | | * 2,000 | | 2,655 | | 3,259 | | 1,680 | | * 1,500 | | * 1,500 | |
| Chili | | 42,916 | | 37,989 | | 42,909 | | 41,099 | | 41,648 | | 38,500 | |
| Cape of Good Hope—Cape Copper Co. | | 5,038 | | 5,087 | | 5,000 | | 5,000 | | 5,000 | | 5,000 | |
| Namaqua Copper Co. | | ... | | ... | | ... | | ... | | ... | | 450 | |
| Canada | | * 50 | | 50 | | 221 | | 329 | | 236 | | ... | |
| United Kingdom | | 3,662 | | 3,875 | | 3,464 | | 2,620 | | 3,350 | | * 3,000 | |
| Germany—Mansfeld | | 9,800 | | 10,909 | | 11,516 | | 12,643 | | 12,582 | | 12,450 | |
| Other German Mines | | 1,000 | | 1,743 | | * 1,800 | | * 2,000 | | * 2,200 | | * 2,800 | |
| Hungary | | * 976 | | 976 | | * 976 | | 680 | | 800 | | * 800 | |
| Italy | | * 1,380 | | * 1,480 | | * 1,400 | | * 1,600 | | 1,325 | | 835 | |
| Japan | | * 3,900 | | * 3,900 | | * 4,800 | | * 7,600 | | * 10,000 | | * 10,000 | |
| Mexico | | * 400 | | 333 | | 401 | | 489 | | 291 | | 375 | |
| Newfoundland—Betts Cove | | * 1,500 | | 1,718 | | 1,500 | | 1,053 | | 668 | | 778 | |
| Norway—Vigsnæs | | 2,040 | | 2,350 | | 2,300 | | 2,340 | | 2,390 | | 2,180 | |
| Other Norwegian Mines | | 386 | | * 290 | | * 290 | | * 290 | | 316 | | 380 | |
| Peru | | * 600 | | 615 | | 440 | | 395 | | 362 | | 229 | |
| Russia | | * 3,081 | | * 3,000 | | * 3,000 | | 4,750 | | 5,000 | | 5,000 | |
| Sweden | | 1,074 | | 995 | | 798 | | 732 | | 662 | | 775 | |
| Spain and Portugal | Rio Tinto | 16,215 | 35,474 | 16.666 | 38,589 | 17,389 | 38,774 | 20,472 | 43,655 | 21,564 | 44,664 | 23,484 | 45,949 |
| | Tharsis | * 9,151 | | * 10,203 | | * 9,000 | | * 9,800 | | * 10,800 | | * 11,500 | |
| | Mason and Barry | * 6,603 | | * 8,170 | | * 8,000 | | * 8,000 | | * 7,500 | | * 7,000 | |
| | Sevilla | 1,705 | | 1,340 | | 1,885 | | 2,026 | | 2,000 | | 1,800 | |
| | Portuguesa | 1,000 | | 1,410 | | 1,700 | | 2,357 | | * 2,300 | | 1,665 | |
| | Poderosa | * 800 | | * 800 | | * 800 | | 1,000 | | * 500 | | * 500 | |
| United States of America | Lake Superior | 22,200 | 25,010 | 24,350 | 30,882 | 25,440 | 40,470 | 26,650 | 51,570 | 30,925 | 64,700 | 32,210 | 74,050 |
| | Montana | 2,810 | | 6,532 | | 4,045 | | 11,010 | | 19,255 | | 30,270 | |
| | Arizona | | | | | 8,030 | | 10,660 | | 11,935 | | 10,135 | |
| | Other States | | | | | 2,955 | | 8,250 | | 2,585 | | 1,435 | |
| Venezuela—New Quebrada | | 1,800 | | 2,823 | | 3,700 | | 4,018 | | 4,600 | | 4,111 | |
| | | 153,057 | | 161,711 | | 177,823 | | 197,936 | | 217,483 | | 221,715 | |

The imports and exports of copper to and from the United Kingdom during the five years ending 1884 were as follows:—

| | 1880. | 1881. | 1882. | 1883. | 1884. |
|---|---|---|---|---|---|
| | Tons. | Tons. | Tons. | Tons. | Tons. |
| Imports . . | 91,667 | 84,190 | 93,279 | 102,857 | 113,610 |
| Exports . . | 59,485 | 61,692 | 55,684 | 59,351 | 64,692 |

## ASSAY OF COPPER ORES.

CORNISH DRY ASSAY.—In an exhaustive paper by M. Moissenet, published in the 'Annales des Mines,'[1] on the English method of assaying copper by the dry way, he very justly remarks that within certain limits this process is not less practical from being somewhat inexact; its object is rather to furnish the smelter with the commercial value of an ore than to indicate the exact amount of copper which it contains. In point of fact, the Cornish assay affords, on a small scale, results similar to those obtained by the smelter on a large one, and any impurities prejudicially affecting the produce in the one case, will equally affect the results in the other.

*Apparatus Employed.*—The furnace employed for copper-assaying in Cornwall is an air-furnace of the form represented in fig. 28, p. 154, and should be about 10 inches long, 9 inches wide, and 14 inches in depth to the grate; in a furnace of the dimensions stated, three fusions for regulus, or four calcinations, may be made at the same time. The fuel is invariably coke; but the size and number of the furnaces used vary in accordance with the requirements of the assayer.

The well-known Cornish crucibles are always employed. They are usually sold in nests of two, and, less frequently, of three. The largest size, about 4 inches in height, is used for calcining ores and for fusions for regulus; the small and middle-sized pots are employed for calcining regulus, fusion for coarse copper, and refining, according to the richness of the ore and the quantity operated on. These crucibles are generally used without covers, and when several assays are being made simultaneously, in order to prevent mistakes, each is marked with a mixture of red oxide of iron and water before being placed in the furnace.

The assayer, in addition to various tongs of convenient shapes for handling red-hot crucibles and removing them from the fire, must be provided with stirring-rods, mould-plates for receiving the fused assays when poured from the crucible, hammers, chisels, an anvil for testing the copper buttons, bronze or cast-iron mortars, an iron slab about 18 inches square for breaking down slags upon, and sieves about 9 inches in diameter with from forty to fifty meshes to the linear inch, for preparing samples. He also requires copper scoops for transferring fluxes, &c., to the crucibles, a regulus-bowl about 10 inches in diameter and 5 inches in depth, kept partially filled with water for cooling the poured assays, having a small annular shelf running round it below the water-level on which the assays to be cooled are placed. Forceps for picking up copper buttons, &c., a ladle for drying samples or washing ores, and flux-spoons for measuring out fluxes are also necessary. The flux-spoon is made of copper, and is usually 1¾ inch in width and ¼ inch deep; a balance capable of turning with $\frac{1}{25}$ grain when laden with 500 grains must be likewise provided.

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[1] Vol. 5$^{e}$ Série xiii. p. 183.

Special weights, of which the unit is termed a *cent*, are used by Cornish assayers for the purpose of facilitating calculation; the system adopted is to divide 400 grains into 100 cents. taken as a standard, the smallest weight being $\frac{1}{16}$, or 0·25 grain. Assays are reported on 100 parts and the unit subdivided into ½, ¼, ⅛, and $\frac{1}{16}$, so that the produce of a sample is stated to be 7¼, 8⅜, 12⅝, 17¾, &c., per cent. The fluxes and reagents used are as follows:—Common salt, dried or fused borax, glass free from lead, lime, fluor-spar, nitre, soda-ash, tartar or cream of tartar, sulphur, charcoal or finely powdered coal, iron pyrites, and white flux for refining.

Refining or white flux, is prepared by deflagrating in a large crucible three parts, by measure, of nitre, two of cream of tartar, and one of common salt; carbonate of sodium, or carbonate of potassium, mixed with a small percentage of nitre, may be used in place of ordinary refining flux.

*Preliminary Examination.* The samples to be assayed usually reach the assayer in a moist state in brown-paper parcels, each weighing about 1½ lb. After drying at a temperature somewhat above 100° C., each sample is ground, sifted, and mixed. If the ore is one which the assayer has not been in the habit of testing, a small portion of it is washed, on a shovel, in an evaporating dish, or in a drying ladle; this is done with a view of ascertaining, approximately, its quality and the proportions of copper, sulphur, arsenic, gangue, &c., it contains. By practice in this manipulation it becomes easy to determine beforehand whether, in the next operation, the ore will or will not require calcination, whether nitre or sulphur should be added, &c. An experienced assayer will in most cases, by a simple inspection of the sample, decide correctly as to the mode of treatment to be adopted. It is consequently only in cases of doubt that washing is resorted to.

*Method of Conducting an Assay.*—The characteristic peculiarity of the Cornish method of assaying is the general preliminary concentration of the copper in the form of regulus. Until within a comparatively recent date this method of treatment was universal, and even rich carbonates and oxides were always assayed on this principle. The relative proportions of the various fluxes employed, as well as the smaller details of manipulation, are varied by different assayers in accordance with the results of their individual experience; but in all cases the Cornish method of assaying comprises the following operations:—

1. Fusion for regulus.
2. Calcination of the regulus.
3. Fusion for coarse copper.
4. Refining.
5. Treatment of the slags for the copper they contain.

1. *Fusion for Regulus.*—The quantity of ore operated on varies in accordance with its richness in copper; 400 grains are commonly used for ores containing under 10 per cent. of copper; 200 grains for ores between 10 and 30 per cent., and 100 grains for samples in which the copper amounts to more than 30 per cent. The fluxes are not weighed, but merely measured in the flux-spoon, their proportions being so adjusted as to yield a fusible slag with the gangue and oxide of iron, resulting from the oxidation of pyrites, &c. They should also produce a slag which separates easily from the regulus, and the amount of nitre, sulphur, &c., should be such as to result in the formation of a regulus containing about 50 per cent. of copper.

Yellow copper ore, without admixture of iron pyrites, contains a larger amount of iron and sulphur than is required to form a regulus of the richness desired.

Vitreous copper ore, on the other hand, requires iron and sulphur, in order to produce a proper regulus. These may be supplied by the addition either of iron pyrites or of a mixture of sulphur and oxide of iron.

In order to obtain from copper pyrites a button of regulus containing about 50 per cent. of copper, it is necessary to oxidize a large portion of the sulphur present. This may be done by a partial roasting ("warming"), by partial roasting and the addition of nitre in the subsequent fusion, or, simply, by the addition of nitre. Either of these methods may be adopted; the first and second require considerable experience with regard to the amount of roasting necessary, but the third is more direct.

Rich oxides and carbonates may be fused directly for coarse copper, care being taken to retain the slags for subsequent treatment; native metal and bar-copper only require refining.

The raw, or more or less calcined ore is intimately mixed with the various fluxes required, introduced into a crucible of the largest size, and over the whole is spread a layer of dried borax. When a preliminary roasting has been resorted to, the crucible employed for that purpose must be preserved for the subsequent fusion. This roasting is conducted in crucibles, which, when placed in the furnace, are packed round with coke to their full height, so that they may be as uniformly heated as possible. A dull-red heat is maintained during the operation, which is continued until the blue flame, due to burning sulphur, ceases, and this usually occupies about ten minutes; if much iron pyrites is present more time will be required. During the first part of the fusion for regulus effervescence takes place from the escape of various gases, but this gradually subsides, until, at the close, the surface of the slag becomes perfectly tranquil. The crucible is now removed from the furnace, and after having received a rotatory motion for the purpose of washing down any particles adhering to the sides, its contents are rapidly poured into an iron mould.

As soon as the slag has solidified, the assay is seized with a pair of copper forceps, dipped two or three times into water, and left to cool on the circular shelf fitted around the inside of the regulus-bowl. This has the effect of fissuring the slag in all directions and causes the regulus to separate easily from it. Should any slag adhere to the button obtained, it will generally be on the upper surface, and may be removed by a slight tap from either a light hammer or the edge of a spatula. After the regulus has been thus carefully separated from the slag, the latter must be examined to see that it contains no enclosed globules of regulus. If any are found, they must be picked out and added to the principal button previously obtained, care being taken to avoid the addition of particles of slag. In order to save time, the regulus is sometimes poured into one cavity of the mould and the slag into another. In order to do this successfully, however, a considerable amount of practice is required; but if there is any doubt of the cleanness of the slag, or the regulus has not been perfectly separated from it, it may be re-melted with the addition of a little sulphur. The button of regulus thus obtained must be added to that previously separated from the slag by pouring.

A good regulus should be reddish-brown in colour, slightly convex on its upper surface, very much fissured, and easily reduced to powder. When the regulus is too *coarse* it is more or less flat and is often vesicular on its upper surface; it is also comparatively hard, and varies in colour from iron-grey to brass-yellow. When a coarse regulus has been obtained there is but little fear of the slag retaining copper, but the calcination of the regulus is not so readily effected. When the ore operated on is very poor, it is sometimes desirable to obtain a coarse regulus in order to insure the complete separation of the copper.

When regulus is too *fine*, the button is more or less spherical, and is smooth, bright, and semi-metallic in appearance. Externally its colour is nearly black, but when freshly broken the fractured surface is of a dark bluish-grey colour, and presents a very compact structure. Such a regulus is more difficult to calcine than one which is not so fine, and there is also in such cases danger of the slags retaining a certain amount of copper.

2. *Calcination of the Regulus.*—The regulus is first reduced to a fine powder in an iron or bronze mortar; after its removal a little coke-dust is rubbed down in the mortar for the purpose of removing the last particles and is added to the powdered regulus. In this finely divided

state the mixture is introduced into one of the smaller or middle-sized crucibles, according to the quantity of regulus to be operated on, and several calcinations are carried on in the furnace at the same time. The furnace is filled with fresh fuel to within a short distance of the top, and the crucibles are arranged upon it with a slight inclination forward, so that air may readily pass over the surface of the powdered regulus. A round stirring-rod of wrought-iron, about $\frac{3}{16}$ths of an inch in diameter, flattened at one end to a chisel-edge, and having a ring turned at the other, is inserted into each crucible; when not held in the hand, these are allowed to lean against a support in order that they may be retained in their positions. The calcination is commenced at a dull-red heat, which is gradually increased to bright redness, in proportion as the contents of the crucible are enabled to bear it, without becoming agglomerated. The time necessary for complete calcination is usually about half an hour; stirring must be constantly kept up during the first fifteen or twenty minutes, after which it need only be occasional. When clotting occurs, the regulus must be removed from the crucible, ground with a little coke-dust, and again calcined; if, however, agglomeration, to any considerable extent, has taken place, it is better to throw away the assay and begin afresh. The calcination is complete when the odour of burning sulphur is no longer evolved, and the sample is then said to have been roasted *sweet*. When this occurs, the crucible and rod are removed from the fire, and when cold, any portion adhering to the rod is carefully scraped off into the crucible. The same crucible is employed for the subsequent fusion. The calcination both of the raw ore and regulus is sometimes conducted in a scorifier heated in a muffle-furnace; calcination is more readily effected by this means and the operation is much expedited, but in Cornwall it is almost universally performed in crucibles.

3. *Fusion for Coarse Copper.*—The flux employed for this operation is usually a mixture of tartar and nitre in such proportions that the tartar is in excess of the amount required to make carbonate of potassium by ignition with the latter. In addition to this some assayers add borax, others use pounded glass, some use neither, while many employ common salt. In the metallurgical laboratory of the Royal School of Mines a mixture of tartar or charcoal with sodium carbonate is employed, but the amount required will of course depend on the weight of calcined regulus to be treated; the addition of an excess will not, however, be attended with serious inconvenience. A mixture of 50 grains of nitre, 180 of tartar, and 36 of borax, is sufficient for the reduction of a calcined regulus weighing, previously to roasting, from 48 to 50 grains. For a button, weighing from 90 to 100 grains, 85 grains of nitre, 220 of tartar, and 50 of borax should be employed. These amounts are not, however, weighed, since, with practice, it becomes easy to measure, with sufficient accuracy, the quantities required.

The calcined regulus is mixed with proper fluxes in the crucible employed for its calcination, and is then introduced into a fire of coke heated to bright redness; fusion takes place in from ten to fifteen minutes, and as soon as the effervescence ceases the melted contents are poured into a mould. When the slag has become set, the assay can be cooled by being dipped into water, or it may be allowed to remain in the mould until sufficiently cold to handle. The slag should be black and glassy, and neither it nor the inside of the crucible should present any streaks or patches of red, due to the presence of copper. These slags are retained for subsequent fusion, although in many cases they are practically free from copper.

4. *Refining.*—The crucible employed in the previous operation is placed well down among the coke in the assay-furnace, in such a position as to be directly under the line of junction of the two bricks forming the cover, and when it has become red hot the button of coarse copper is dropped into it. The furnace is now nearly closed, and the operation closely watched through the opening between the bricks. Fusion is soon effected and a slight evolution of gas takes place from the dull surface of the metal. After the expiration of a short time the film of oxide begins rapidly to disappear from the surface of the button, which becomes perfectly bright at the edges, and reflects a bluish-green light from its centre, producing the appearance technically known as the "eye" or "star." Some refining flux,[1] or refining flux and salt, previously placed in a copper scoop ready for immediate use, is now introduced upon the top of the fused button, and the furnace is again closed. In about two minutes after the introduction of the flux the crucible is withdrawn from the fire and its contents rapidly poured into a mould. When it has sufficiently set, the button, which is covered with slag, is removed between the jaws of a pair of forceps and held with its lower side beneath the surface of the water in the regulus-pan; by this means the slag is easily detached. The whole operation of refining does not usually occupy above seven minutes. The button of copper when *fine* is nearly flat and has its upper surface coated with a thin film of an orange-red colour. The metal in this condition is soft, malleable, and tough, breaking with difficulty, and presenting a closely fibrous fracture. It is, however, more commonly somewhat *dry*, presenting a slight depression on its upper surface, and when broken exhibiting a granular fracture which has a purple tint. When not sufficiently refined, the button, both externally and when broken, presents, to a certain extent, the appearance of coarse copper; in this case refining must be repeated. Salt is generally used in refining, as it not only checks the too rapid action of the refining flux, but also probably aids in the separation of antimony, arsenic, &c.

5. *Treatment of the Slags for Copper.*—The slags resulting from the operations of reducing and refining are subsequently treated by fluxing with a couple of spoonfuls of tartar or a little powdered charcoal. By this means the copper retained by the slags will assume the form of a

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[1] Made by igniting together three volumes of tartar, two volumes of nitre, and a small quantity of salt.

small button or *prill*, the weight of which must be added to that of the principal button. The prill obtained usually weighs from 1 to 5 grains, according to the nature of the assay and the skill of the operator. Cornish assayers refine the copper thus obtained from re-melting the slags, but the quantity is often so small that the error resulting from omitting this operation would practically be unimportant.

German Method of Assaying.—The method of conducting copper assays in some of the smelting establishments of Central Europe differs in several particulars from that adopted in this country, and the results obtained are stated to be somewhat higher.

The apparatus employed consists of an ordinary muffle-furnace; small egg-shaped crucibles provided with a foot; scorifiers of fire-clay about 2½ inches in diameter, and an assortment of tongs, hammers, &c. In addition to borax, salt, glass, powdered charcoal, and graphite, metallic lead is also employed; black flux, prepared by deflagrating a mixture of two parts of crude tartar and one of nitre, is used as the reducing agent.

The process includes the three following operations:—

1. Roasting; calcining.
2. Melting for coarse copper.
3. Refining.

1. *Roasting; Calcining.*—About 4 grammes of dry ore are weighed out, mixed with one gramme of powdered graphite, and spread upon the bottom of a scorifier. This is introduced into a heated muffle and is almost continuously stirred during fifteen or twenty minutes, after the lapse of which time sulphurous fumes should be no longer given off. The scorifier is then removed from the muffle and allowed to cool, the assay carefully brushed from it into a bronze or cast-iron mortar, where it is finely ground, and, after being again mixed with 1 gramme of pulverized graphite, it is subjected to a second calcination similar to the first. At the expiration of about fifteen minutes the mass will have generally assumed a reddish-brown appearance, and the evolution of sulphurous fumes will be no longer perceived.

When either lead or antimony is present in an ore, the roasting requires to be conducted with considerable care, since if the assay were too strongly heated its surface would become fused, and its further calcination materially interfered with.

2. *Melting for Coarse Copper.*—After roasting, the metals in the assay will be principally in the state of oxides, and the object of the fusion, which now follows, is to collect the whole of the copper in the metallic form, while the principal portion of the metals with which it is associated passes into the slag as silicates. The calcined ore is carefully removed from the scorifier or roasting-dish, and is mixed in a mortar with from 3 to 3½ grammes of black flux; this mixture is introduced into the crucible, and upon it are placed, without mixing, 8 or 9 additional grammes of black flux; on this are placed 1½ to 2 grammes of powdered

glass and 1 gramme of borax. Instead of black flux, a mixture of one hundred parts of carbonate of potassium with from ten to twelve parts of wheaten flour may be employed. The whole is covered by a layer of from 8 to 12 grammes of common salt, and lastly, a piece of charcoal of the size of an ordinary bean is added. The crucible is now covered and placed in the muffle, where it is gradually raised to a white heat, the fusion being completed in about thirty or thirty-five minutes. When complete fusion has been effected, and the slag is in a perfectly liquid state, the crucible is withdrawn from the muffle, and, after being allowed to cool, is broken and the button of metal extracted. This must not be covered by a crust of sulphides, and the slag should be glassy, and of a dark-green colour, without any traces of red.

3. *Refining.*—As in the case of the Cornish assay, this process has for its object the removal from the copper of the various other metals by which it is contaminated. In order to effect this, advantage is taken of the property possessed by copper of remaining, practically, unchanged when exposed in a fused state to the action of a current of air, so long as a more oxidizable metal is present. The metals thus oxidized in the presence of borax, which is added for that purpose, are taken up and carried off as a fusible slag. The scorifier employed for refining the coarse copper often consists of a fragment broken from the side of a pot in which a fusion has been already effected; in this, which has been previously heated to bright redness in the muffle, is placed the button of copper to be refined, wrapped in paper with its own weight of borax.

As soon as the copper shows a convex perfectly clear surface, and is surrounded by a thinly fluid ring of borax, the mouth of the muffle is opened, and a current of air allowed to play over its surface. If the surface of the copper be not clear, but is covered with a black coating, at the time the muffle is at a white heat, a further addition of borax must be made. Should this not result in the production of a bright surface, a small piece of lead must be dropped into the scorifier and the heat of the furnace increased to its maximum. When the button of coarse copper is very impure and does not contain much above one-half its weight of pure copper, it must be first placed on the scorifier with borax only, the lead being added towards the close of the operation. A small portion of the lead thus added escapes in the form of fume, while the greater part passes into the slags. Arsenic is to a very great extent volatilized, but a portion is retained in the slag. The removal of nickel by scorification is extremely difficult and necessitates a large addition of lead, which results in a loss of copper.

When the copper has become fine it *brightens* like silver, but less distinctly. Care must be taken that the temperature of the muffle at the moment of brightening does not much exceed that at which pure copper solidifies. The assay, which in brightening exhibits a peculiar greenish light, is now removed from the furnace, cooled, quenched in water, freed from slag, and weighed. A good assay button is exteriorly of a pure copper-colour, is ductile, uniformly granular, and rose-red in the fracture. When a button has not been sufficiently refined it is externally red, but its fracture is grey; an over-refined button is dark red on the surface and brittle, the fracture being rather smooth than granular.

Refining on the cupel is in use in some of the smelting-works of the Harz; it is not more accurate than the above, although perhaps more suitable for copper containing large quantities of lead.

**Wet Assay of Copper Ores.—Precipitation by Metallic Zinc or Iron.**—This method of estimating copper is especially adapted for ores containing little or no arsenic, and consists in attacking the mineral to be examined by a mixture of nitric and hydrochloric acids, the subsequent expulsion of the nitric acid, and lastly, the precipitation of the copper from its chloride, by metallic zinc or iron.

The mineral to be operated on must be first ground and passed through a fine sieve. Of this powder, 100 grains are weighed and introduced into a narrow-necked flask of German glass. Nitric acid is now cautiously added, and the flask gently warmed on a sand-bath; since if it were too suddenly heated, or too large a quantity of acid were added at a time, violent ebullition might ensue, and a loss on the assay be the result.

When the evolution of nitrous vapours entirely ceases, or they become much diminished in quantity, add gradually hydrochloric acid, place the flask in an inclined position on the sand-bath, and cause its contents to boil gently. This must be continued until the residue, if any remain, appears to be free from metallic stains.

The contents of the flask must now be carefully transferred to a porcelain dish and evaporated to dryness, with the usual precautions. When sufficiently cool, moisten the residue with hydrochloric acid, heat gently, and afterwards add water, boil, and filter into a beaker.

A piece of zinc or polished wrought-iron, about 2 inches in length, ¾ inch in width, and ¼ inch in thickness, is now attached to a string and lowered to the bottom of the beaker. It is essential to the success of this operation that the whole surface of the metal should be completely covered by the liquid, otherwise a portion of the precipitate would become oxidized and the results vitiated. The contents of the beaker must now be kept in gentle ebullition until the whole of the copper present has been thrown down, which is ascertained by the liquor becoming colourless. This may be confirmed by trying a drop of the liquid on the surface of a piece of clean sheet-zinc, or by the blue colour produced by the addition of ammonia in excess to solutions containing copper.

After having ascertained that the whole of the copper has been thrown down, carefully clean with a feather the piece of metal which has been used as a precipitant, and then decant off the supernatant liquor by the aid of a small glass syphon, and repeatedly wash with warm water, until the precipitated copper is entirely free from any traces of chloride of zinc or chloride of iron.

The washing water is finally decanted, leaving the precipitated copper in the bottom of the beaker, which is now placed in a water-bath or in a warm place near a furnace, until it has become completely dried.

In this operation it is necessary to so regulate the heat as to prevent the oxidation of the copper, by which the accuracy of the result would be impaired.

The copper thus obtained is subsequently brushed into a watch-glass, by the aid of a camel-hair brush, and weighed; on deducting from this weight the tare of the watch-glass, the result represents the percentage of copper present.

When the mineral operated on contains either lead or antimony, no appreciable trace of these metals will be found in the copper precipitated. If large quantities of lead be present, it is, however, best to add sulphuric acid or sodium sulphate, and to filter previous to the precipitation of the copper.

If no perceptible oxidation of the precipitated copper has taken place, its weight will sufficiently indicate the produce of the ore; it is, however, safer to check the result so obtained by converting the metallic copper into cupric oxide, from the weight of which the yield of the ore is readily calculated. The conversion of the finely divided copper into cupric oxide may be effected by exposing it to a red heat, in an uncovered porcelain crucible, until its weight becomes constant; or the copper may be transformed into nitrate by the addition of a few drops of nitric acid, and cupric oxide obtained by subsequent ignition. As this oxide is highly hygroscopic it must be weighed rapidly, and while still warm.

More accurate results may, however, be obtained by dissolving the precipitated copper, and estimating its amount by one of the following volumetric methods.

By Potassium Cyanide.—This method of estimating copper was first made known by Mr. Henry Parkes in 1851, and is one of the most convenient and accurate of the various processes for assaying copper ores by means of standard solutions. This process depends on the circumstance that when cyanide of potassium is added to a blue ammoniacal solution of copper, the latter gradually loses its colour and finally becomes colourless. The amount of cyanide necessary to discharge the whole of the colour from an ammoniacal solution is, all other circumstances being the same, directly in proportion to the quantity of copper present; it is consequently easy, by means of comparative experiments, to establish a standard by which the amount of copper in a given weight of ore may be determined.

The only apparatus required is an ordinary Mohr's burette of 50 c.c. capacity and 18 inches in length; this is supported vertically by a wooden stand, which admits of its being either raised or lowered by means of arms with screws sliding upon an upright pillar. A great number of assays may be conducted at one time by the same person by the aid of a series of such burettes arranged on a stand and supplied with the same titrated solution. Where a large number of assays have to be made daily, the burettes may be connected with a large stoneware or glass jar, supported on a convenient shelf, containing the titrated solution, with which they are filled by syphons connected with the bottom of each, by glass T-pieces and indiarubber tubes provided with spring clips.

The best cyanide for this purpose is that known as photographic cyanide, as solutions prepared from it may be kept a long time without becoming either discoloured or muddy. To prepare a standard solution, 260 grammes of photographic cyanide may be dissolved in 4 litres of distilled water; this liquid should be kept in green-glass bottles free from lead. The solution is standardized as follows: about 1 gramme of chemically pure and perfectly clean copper is dissolved in dilute nitric acid, and the solution boiled until all nitrous fumes have been expelled; it is then diluted with water, and ammonia in excess added. The blue solution thus obtained is made up to 750 c.c. and divided into three equal portions of 250 c.c. each. The burette is now filled to the level of the uppermost division with the standard solution of cyanide of potassium, and as soon as the copper solution has become quite cold, the beaker containing it is placed under the burette, and the cyanide of potassium is run into it in small quantities at a time, care being taken towards the close to avoid the addition of the smallest quantity more than is necessary. The cyanide solution is finally introduced by successive small additions until the blue colour has been completely discharged, and has been replaced by a very faint tint of violet. The number of divisions necessary for the decoloration is now read off and noted, and the second and third portions of the copper solution proceeded with in the same manner. The mean of the three results is taken, and from it is calculated the amount of copper corresponding to each c.c. of the cyanide solution used. With the proportions above specified it will be found that about 145 c.c. are equal to 1 gramme of copper. The above is the average strength of the solution employed in the various copper-works in which that metal is extracted by the wet process from burnt Spanish pyrites. For the assay of richer ores a standard solution of twice the above strength may be conveniently employed. For practical purposes the standard will not require to be checked more frequently than once a week.

In order to make an assay by this process, a weighed quantity of copper ore may be placed in a flask, moistened with sulphuric acid, and nitric acid added. The whole is now digested at a gentle heat, with the occasional addition of nitric acid, until coloured nitrous fumes are no longer evolved. As soon as the ore has been completely decomposed, the contents of the flask are transferred, without filtration, to a beaker of convenient size, diluted with distilled water to about 300 c.c., and excess of ammonia added. The intensely blue solution thus obtained is allowed to become cold, and the separated ferric hydrate falls to the bottom, where the insoluble gangue has already collected. Without separating these, the standard solution of potassium cyanide is gradually and cautiously added, with occasional stirring of the solution, until the blue colour has entirely disappeared and has been replaced by a faint violet tint. The number of divisions necessary to produce this effect are read off, and from the quantity of solution employed the percentage of copper contained in the ore is calculated. The method of doing this will be readily understood by the aid of the following example:—

145 divisions of the burette equal 1 gramme of copper; 2 grammes of copper ore require 30 divisions for decoloration; consequently

$$\begin{array}{ccccccc} \text{Divisions.} & & \text{Divisions.} & & \text{Copper.} & & \text{Copper.} \\ 145 & : & 30 & :: & 1{\cdot}0 & : & 0{\cdot}2069 \end{array}$$

$$0{\cdot}2069 \times \frac{100}{2} = 10{\cdot}345 \text{ per cent.}$$

When a sulphurous ore is operated on, it will, in the majority of cases, be completely oxidized by a mixture of sulphuric and nitric acids; but should any globules of sulphur remain, they may be taken out after the dilution of the solution, ignited, and the residue attacked by nitric acid and added to the copper already dissolved. The attack of some varieties of ore is best made by aqua regia.

Owing to the influence exercised by varying quantities of ammonia and of ammonium salts upon the decoloration of copper solutions by potassium cyanide, it is necessary that both the test solution, originally prepared, and the various cupreous solutions subsequently assayed, should contain, as nearly as possible, equal amounts of ammonia. The presence of ferric hydrate imparts a greenish tint to the ammoniacal solution, and its proper shade is best observed by placing the eye on a level with the top of the liquid.

*Iron.*—This metal, in the state of ferric hydrate, does not interfere with the results, excepting that it takes some time to settle after the stirring which accompanies each addition of the standard solution; its effect is consequently only to slightly increase the time occupied in making an assay.

*Lead* and *bismuth* are, likewise, without effect upon the result.

*Arsenic* does not interfere with the results excepting in the presence of iron, when it forms an arsenate giving rise to a brownish tinge in the liquid. The removal of the arsenic may be effected by adding magnesium sulphate in excess. As soon as a precipitate is no longer formed, and the solution has acquired its characteristic blue colour, the assay may be proceeded with in the usual manner.

*Manganese* is not often found in copper ores in sufficient quantities to materially affect the results. When present, it may be completely removed by adding to the ammoniacal solution sodium carbonate, with a few drops of bromine, and boiling; it will thus be precipitated as manganic oxide, and when the cupreous solution has become cold the assay may be proceeded with as though the ore had not contained manganese.

*Silver.*—Should this metal be contained in the ore in such quantity as to exercise, practically, any influence on the assay, it may be removed by adding a few drops of hydrochloric acid to the solution, and filtering before the addition of ammonia; it is evident that when hydrochloric acid has been used for the attack, silver cannot exist in notable quantity in the resulting liquor after dilution.

*Zinc, Nickel, and Cobalt.*—These metals would, if present, render the results obtained utterly unreliable, and in such cases the copper must be first separated by precipitation. This may be effected by means of a piece of either zinc or iron, in the way already described, care being taken that nitric acid is not present. The precipitate thus obtained is subsequently dissolved in nitric acid, and the amount of copper present determined by the cyanide solution in the usual way. Instead of precipitating the copper in the metallic state, it may be thrown down as sulphide by sulphuretted hydrogen, and the sulphide re-dissolved and subsequently estimated by potassium cyanide. Sodium hyposulphite may also be employed as the precipitant.

BY A STANDARD SOLUTION OF SODIUM HYPOSULPHITE.—This process, which has long been employed at the Royal Arsenal, Woolwich, for the estimation of copper in various alloys, was first described by Mr. E. O. Brown.[1] It is especially adapted for the estimation of copper in various descriptions of commercial copper and bronze, in which lead and iron are not present in large quantities. This process is founded on the reaction taking place between iodine and hyposulphurous acid, which results in the production of hydriodic and tetrathionic acids. The completion of the reaction is rendered manifest by the bleaching effect exercised on a solution of starch added during the progress of the operation. Slight differences in temperature or variations in the details of manipulation do not materially affect the results.

The following reagents are required:—

1. A solution of sodium hyposulphite, prepared by dissolving 1,000 grains of the re-crystallized salt in a Winchester bottle of distilled water, and standardizing with pure electrotype copper, the mean result being taken as the correct standard.

2. Potassium iodide, free from potassium iodates; this may be used in crystals.

3. A solution of starch, made by boiling starch in a large quantity of water, allowing it to stand until the insoluble residue has subsided, and pouring off the clear supernatant liquor for use.

To make an estimation of copper by this process, from 6 to 8 grains of the metal or alloy to be examined are dissolved in dilute nitric acid, and nitrous acid expelled by boiling. To this solution, diluted with a small quantity of water, sodium carbonate is gradually added, until a certain portion of the copper remains precipitated. An excess of pure acetic acid is then added, and the whole transferred to a flask of about a pint capacity. About 60 grains of potassium iodide are now introduced, and allowed to dissolve. When this has become completely dissolved, the contents of the flask are shaken; the standard solution of sodium hyposulphite is gradually run into it from a burette, until the chief portion of the free iodine has been removed, and the liquid has assumed a yellow colour. At this stage a small quantity of starch solution is added, and the introduction of sodium hyposulphite continued until the solution becomes colourless. The number of divisions are now read off from the burette, and from the quantity of standard solution employed the amount of copper is calculated.

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[1] 'Quart. Jour. Chemical Society,' April 1857; Percy, 'Metallurgy of Copper,' p. 486.

For copper ores, cupriferous pyrites, burnt ores, &c., this process should be slightly modified. From 50 to 150 grains of the material to be operated upon are dissolved in nitro-hydrochloric acid, evaporated to dryness with excess of sulphuric acid, diluted with water and filtered. From this solution the copper is precipitated either as $CuS$, by hydrogen sulphide, or as $Cu_2S$, by sodium hyposulphite, the precipitate dried, ignited, dissolved in nitric acid, evaporated with excess of sulphuric acid, and diluted with water to remove any trace of lead. To the filtered solution sodium carbonate in excess is added, and subsequently acetic acid to acid reaction. Potassium iodide is now introduced, and the amount of copper present calculated in the usual way.

This process affords exceedingly accurate results, and is much employed for the assay of cupriferous pyrites.

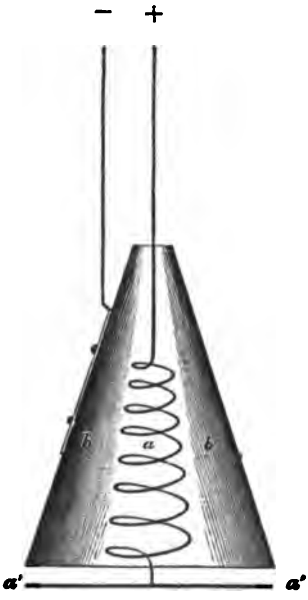

Fig. 117.—Spiral and Cone.

Assay by Electrolysis.—In order to make a copper assay by this process, from 2 to 5 grammes of the substance to be operated upon, according to its richness, should be finely pulverized and then attacked by aqua regia. When effervescence has subsided, add from 4 to 5 c.c. of sulphuric acid diluted with an equal volume of water; evaporate to complete dryness and ignite. The sulphur will be thus burnt off, and the residue is dissolved by the addition of 20 c.c. of weak nitric acid and boiling, the liquid being so diluted as to measure about 300 c.c. Two platinum electrodes are now suspended in the filtered solution, namely, the spiral, *a*, fig. 117, in connection with the positive pole, and the cone, *b*, connected with the negative pole, either of a Clamond thermo-electric battery, or of some other apparatus capable of yielding a continuous but feeble current of electricity. The coil of platinum wire, *a*, terminates in a platinum ring, *a′*, supported by three radial wires, fig. 118, and must be so placed as to occupy a central position within the cone. Contact between any portion of the spiral or ring with the platinum cone would necessarily prevent a deposit of copper from taking place.

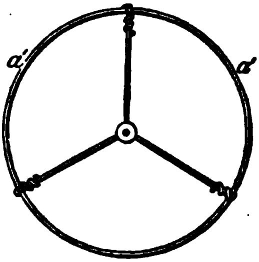

Fig. 118.—Platinum Ring.

The upper part of the platinum cone should project slightly above the surface of the liquid, and when the thermo-pile of Clamond is employed, complete deposition of the copper will be effected in about six hours. The operator will consequently prepare his solutions and place them, severally, in connection with a suitable battery until the whole of the copper has been deposited, when nothing remains to be done beyond washing, drying, and weighing the copper.

In this way six or more different assays may be made at the same time, all the spirals being placed in communication with the positive pole of the battery and all the cones connected with its negative pole.

In order to be quite certain that the whole of the copper has been deposited upon the platinum cones, they must, before being withdrawn, be lowered until they are completely immersed in the liquid. If after the expiration of half an hour no coating of copper has been deposited on the newly submerged surface, it may be concluded that the whole of that metal has been thrown down.

When the operation is terminated, the platinum cones are, one after another, carefully removed, care being taken to preserve, as far as possible, connection with the battery while they are being severally withdrawn from the liquid; any loss that might result from the action of free acid upon so large a surface of copper being thus obviated.

Each cone, upon its withdrawal from the liquid, is first well washed with distilled water, dried, and weighed. As the exact weight of each cone was accurately determined before the operation, the difference between the two weighings gives the weight of copper.

If the material to be examined is an alloy containing zinc, the copper is completely deposited by electrolysis while the zinc remains in solution.

When arsenic and antimony are present, they are precipitated in the metallic state upon the negative electrode in the form of a black deposit, which does not begin to appear until the whole of the copper has been thrown down. If either of these metals is present, the ore should be carefully roasted before being attacked by acid, and the cone must be removed from the solution on the first appearance of a dark shade over its surface.

When lead is present in a solution, it is deposited chiefly in the form of plumbic dioxide on the positive pole; but its estimation by electrolysis cannot be correctly effected, since the composition of the deposit obtained is not sufficiently constant. Instead of using a thermo-pile, any sufficiently constant battery or a properly arranged dynamo may be employed.

## METALLURGY OF COPPER.

A considerable proportion of the ores of this metal consist of mixtures of various sulphides, and all the ordinary methods employed for their metallurgical treatment are largely dependent on the relative affinity for sulphur and oxygen possessed respectively by copper and the different metals with which it is associated. Copper, at high temperatures, has a stronger affinity for sulphur than iron, which is the metal with which it is most plentifully found, possesses for the same body. Iron, in the presence of sulphur and oxygen, appropriates the latter, and, uniting with silica, forms a liquid slag; while copper and sulphur, combining, give rise to a fusible sulphide known as regulus or matte. In almost all cases, therefore, the treatment of copper ores consists in a system of alternate roastings[1] or calcinations and fusions, by which iron is gradually removed as silicate, while copper is progressively concentrated in a series of sulphides, gradually increasing in richness.

Any oxide of copper that may result from the process of calcination is, during the subsequent fusion, converted into sulphide at the expense of sulphide of iron; silicates of that metal and copper regulus being the result. The siliceous slag thus carries off the larger portion of the iron originally present in the ore, while the regulus contains, practically, the whole of the copper in a concentrated form. This product is again subjected to further calcination, and afterwards fused with siliceous matter, preferably associated with natural oxides or carbonates of copper. Another slag is thus obtained at the expense of the chief part of the iron retained in the first regulus, while the regulus now produced is much richer in copper than that resulting from the first fusion. Similar operations are repeated until impure metallic copper and a liquid slag is produced, with, in some cases, a small quantity of very rich regulus. Any regulus thus obtained is subsequently added to that produced from similar operations, and consequently the final result will, in all cases, be impure metallic copper. This is afterwards rendered ductile and malleable by a process of refining.

The processes by which copper is obtained from its ores by smelting vary in different localities, and it would consequently be impossible to describe more than a very limited number at such length as to render them intelligible. We shall therefore confine ourselves to a general description of the most important methods, each of which may be regarded as typical of the class to which it belongs, and to a new and characteristic process, by which copper mattes are rapidly reduced to the state of blister-copper. The examples chosen for this purpose are :—

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[1] Copper-smelters in this country make a distinction between calcination and roasting, the latter term being by them exclusively applied to operations of the character of No. V. (p. 418). This difference is recognised when treating of the Welsh method of copper-smelting, but in all other portions of this volume the terms are employed as synonymous.

1. The Welsh method of copper-smelting, as formerly conducted in South Wales and Lancashire.

2. The method applied to the treatment of cupriferous schists at Mansfeld, Germany.

3. The Manhès method of copper-smelting, as carried out in France.

The first, in a modified form, is employed for a large proportion of the copper-production of the world, and is specially adapted for securing regularity of yield and the best commercial results from ores of very varying percentage and composition.

The second is used in the treatment of an ore which, although exceedingly poor in copper, occurs in very large quantities, contains a notable amount of silver, and never varies materially in composition. The various processes employed at Mansfeld, including those by the aid of which the extraction of silver is effected, are the result of an experience acquired during a long series of years by a succession of carefully trained superintendents, and the whole system has been thus brought to a high degree of efficiency.

The third method of copper-smelting consists in smelting for matte in a blast-furnace, subjecting the re-fused sulphides to a blast of compressed air in a modified Bessemer converter, and refining the copper produced in the usual way.

## WELSH METHOD OF COPPER-SMELTING.

The ores suitable for treatment by this method may be classified as follows :—

1. Copper pyrites, with iron pyrites, unmixed with either oxide or carbonate of copper ; the gangue is usually siliceous.

2. Mixtures of various sulphides containing less iron pyrites than the above, with small proportions of native metal, and of the oxides and carbonates of copper ; gangue generally quartzose.

3. Chiefly oxidized ores, containing inconsiderable quantities of sulphur ; these, for the most part, consist of a mixture of oxides and carbonates with a little native copper, and are usually accompanied by a siliceous gangue.

The processes of copper-smelting are varied, in accordance with the nature of the ores to be treated and the quality of the copper it is desired to produce ; it now seldom comprehends more than *six distinct operations*, but sometimes less.

Under ordinary circumstances the following general conditions should be observed in making up the working mixtures :—

*a.* The mixture of ores operated on (Classes 1 and 2) should not contain less than 9 or more than 15 per cent. of copper ; if poorer than the one, the amount of fuel consumed will be excessive, and if richer than the other, it will be difficult to obtain clean slags.

*b.* The furnace mixture should, after calcination, fuse readily, and afford a clean slag without the addition of any kind of flux.

*c.* The matte, or coarse-metal resulting from the fusion, after calcination, of the furnace mixture, should contain from 35 to 39 per cent. of copper.

In making up the working mixture, oxides and carbonates are not generally used, but they are subsequently employed at various stages of the treatment.

It would be impossible to lay down any invariable rule to be followed in copper-smelting, as the ores and other cupriferous materials operated on by the smelter vary so considerably in character that his operations must, to a great extent, be guided by his personal judgment and experience.

Neglecting small quantities of various substances exercising no material influence on the ultimate results, the average composition of the ores smelted, some years ago, at one of the Swansea works was, according to Napier, as follows:—

| | |
|---|---|
| Cu | 13 |
| Fe | 29 |
| S | 24 |
| $SiO_2$ | 34 |
| | 100 |

Of late years the very large quantities of foreign ores, copper-precipitate, and rich American mattes which have come into the market, coupled with the greatly reduced production of Cornish ores, has materially affected the details of copper-smelting in this country. Since, by smelting together copper-precipitates and sulphurous ores, mattes of any degree of richness can be produced, the necessity for calcination has to a large extent disappeared, and there are now copper-works of considerable importance entirely without calciners.

These changes in the nature of the material to be treated tend rather to simplify than to complicate the process of copper-smelting; but it will, nevertheless, be desirable to describe, in their proper sequence, the typical operations constituting the Welsh process, although certain of them are now often omitted.

The six distinct processes constituting the old Welsh method of copper-smelting are the following:—

I. Calcination of mixed ores.

II. Fusion of calcined ores and *metal-slag* from No. IV. Products, *coarse-metal*, and *ore-furnace slag*, mostly thrown away.

III. Calcination of crushed coarse-metal.

IV. Fusion of calcined coarse-metal with ores belonging to Class 3, and slags from operations V. and VI. Products, *white-metal* or *pimple-metal*, containing from 76 to 79 per cent. of copper, and *metal-slag*, melted in operation II.

V. Roasting the white- or pimple-metal. Products, *blister-copper*, containing about 99 per cent. of copper, and *roaster-slag*, added to charge in operation IV.

VI. Refining and toughening. Products, *marketable copper*, and *refinery-slag*, added to charge in operation IV.

Reverberatory furnaces are exclusively employed in the Welsh method of smelting. These are of two kinds: *calciners* and *melting-furnaces*.

I. *Calcination of Mixed Ores.*—This operation is conducted in a reverberatory furnace, one of the forms of which is represented in figs. 119 and 120; the first being a longitudinal, and the second a horizontal section, above the fire-bridge, on the line A, B, fig. 119. The hearth, R, which is 16 feet in length by about 12 feet 6 inches in width, is formed of fire-brick grouted with fire-clay. The arch descends rapidly from the fire-place, F, to the apertures, H, by which the gases generated during the oxidation of the ore, together with the products of combustion, pass into a flue in connection with a high chimney. Air is admitted, either by an aperture, *d*, which may be partially or entirely closed, or by openings in the fire-bridge, which is sometimes traversed longitudinally by a channel communicating with the atmosphere. Some of the more modern calciners are considerably longer in proportion to their width than that shown in the woodcut, and have a false arch or screen extending a few feet from the fire-bridge in order to protect the ore in that part of the furnace from becoming too highly heated. In other cases, calcination is effected in a close-furnace or muffle, and the sulphur utilized for the manufacture of sulphuric acid. Furnaces of this kind employed in copper-works near St. Helens are fired with gas and provided with Siemens' regenerative apparatus; the results are stated to be satisfactory.

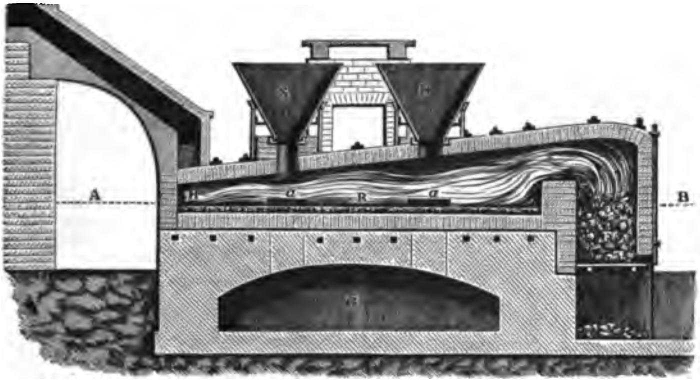

Fig. 119.—Calcining-Furnace; longitudinal section.

The ordinary calcining-furnace, whatever may be its form, is provided with rectangular openings or doors, *a*, immediately behind which are, usually, openings, *e*, in the hearth. During the time the furnace is at work these holes are closed by plates, which are removed at the close of each operation in order to allow of the roasted charge being raked into chambers, C, situated beneath. Iron bearers above the arch of the furnace support two cast-iron hoppers, S, in which is placed the ore to undergo the process of calcination. These are provided with sliding doors, by the withdrawal of which the ore may be caused to fall directly upon the bottom of the calciner.

In the smelting-works in the neighbourhood of Swansea clinker is generally allowed to accumulate in the fire-place so as to form a bed of considerable thickness, and upon this, which is kept sufficiently open to allow of the passage through it of the necessary amount of air to sustain combustion, small coal is burnt. The fuel employed is often a mixture of free-burning and binding coals, in the proportion of about two parts of the former to one of the latter.

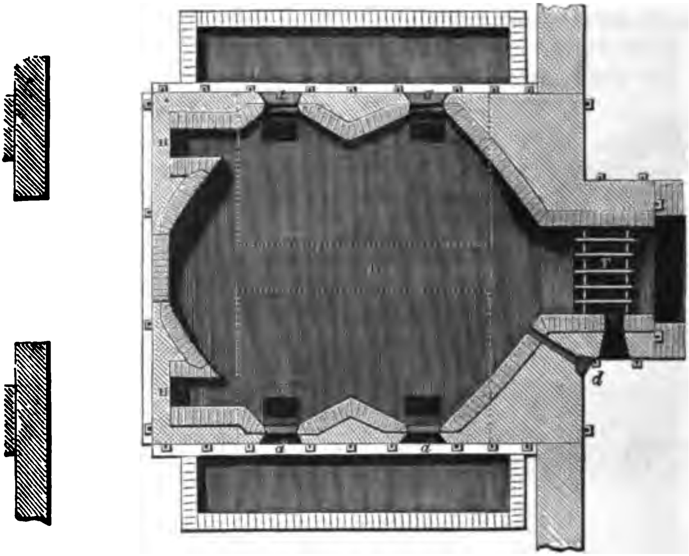

Fig. 120.—Calcining-Furnace; section on A, B.

When the clinkers have become raised to a sufficient height from the bars, their further increase is prevented by occasionally causing the fall of the lower portions by the use of a long iron bar. In this way is formed, throughout the mass, a sufficient number of channels to yield a free passage to the air necessary for combustion, which, in passing through the interstices of the heated clinker, acquires a considerable elevation of temperature; these apertures, although sufficiently numerous for the passage of air, are too small to allow the fine coal or *slack* to descend into the space beneath. The oxygen of the air, which has become highly heated by passing through the bed of clinkers, is, on traversing the fuel, principally converted into carbonic oxide, which, together with nitrogen and sundry products of distillation, passes over the fire-bridge into the furnace. Here the gas takes fire, and is consumed by the air entering either through the opening, *d*, or through the fire-bridge, as well as by various holes left in the doors, by which the lateral openings are closed during the calcination. In this way the whole internal cavity of the furnace is constantly occupied by a sheet of flame, caused by ignited carbonic oxide, which burns on coming into contact with a stratum of atmospheric air so admitted as to spread immediately over the surface of the hearth. The ore is consequently exposed to a current of air, above which is a parallel sheet of burning carbonic oxide, which is inflamed where it comes in contact with the oxidizing stratum, and thereby affords the heat necessary to carry on the operation.

The working of a charge of ore commences without any interval in the action of the furnace, and is begun immediately after the withdrawal of the calcined ores resulting from the preceding operation. The charge varies in weight from 3 to 4 tons, and is introduced by withdrawing the dampers from the bottom of the hoppers, in which it is placed during the working of the preceding charge. As soon as it has been let down, it is spread evenly over the surface of the hearth by long iron rakes, introduced through each of the working-doors, which are closed when the bottom of the furnace has been properly covered. After the expiration of two hours the doors are again removed, and the ore is stirred with iron paddles, in order to expose new surfaces to oxidizing influences.

This operation is repeated at intervals, and after the expiration of from eighteen to twenty-four hours the calcination is sufficiently advanced. In order to withdraw the charge, the workmen open the working-doors, *a*, and after having removed the plates covering the openings, *e*, they draw the ore through the apertures with iron rakes, and cause it to fall into the arched chambers, C, from which, when sufficiently cooled, it is removed and charged into the melting-furnace employed for the next operation.

II. *Fusion of Calcined Ores with Raw Ores, Slags, &c.*—The furnace in which this operation is conducted is represented in figs. 121, 122, and has a hearth, A, about 14 feet long by 11 broad; the first of these figs. is a longitudinal, and the second a horizontal section. The fuel employed in South Wales often consists of a mixture of free-burning and caking coal, consumed on an open fire-grate, F. In Lancashire, free-burning coal only is used. Sand is made use of for the bottom of this furnace, which is so lowered at the part B as to afford a sort of internal basin. To form a charge, calcined ore, from operation I., is fused with slags from operation IV., the products obtained being a regulus known as *coarse-metal*, containing from 35 to 39 per cent. of copper, and *ore-furnace slag*, which is thrown away. After stopping the tap-hole, *a*, with a mixture of clay and sand, a charge weighing about 2½ tons, frequently composed of about two-thirds of calcined ore and one-third of raw ore of Class 2, is let down into the furnace through the hopper, H, and spread evenly over the bottom. The slag is thrown in through the door, *d*, after which the openings are closed and the fire is made up. The operation in this furnace is usually effected in from six to seven hours, when the fused mass, consisting of melted regulus and slag, is well stirred; after this, the latter is skimmed off and raked through the door, *d*, at the end opposite the fire-place, whence it falls into a series of open sand-moulds, M, connected by a central channel, where it assumes the form of nearly rectangular blocks. The furnace is now again charged with a mixture of calcined ores and slags, and the opera-

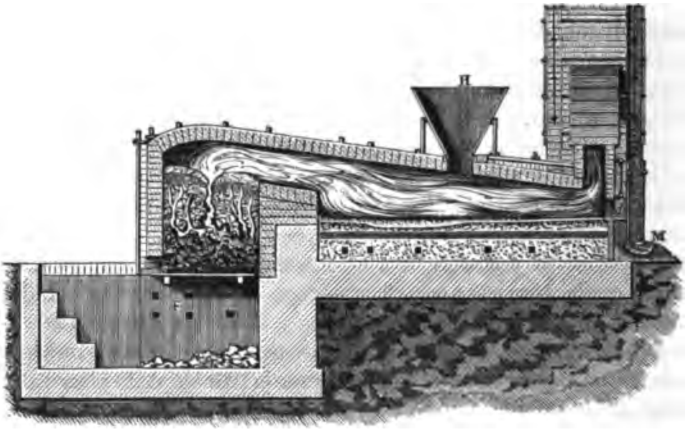

Fig. 121.—Melting-Furnace; longitudinal section.

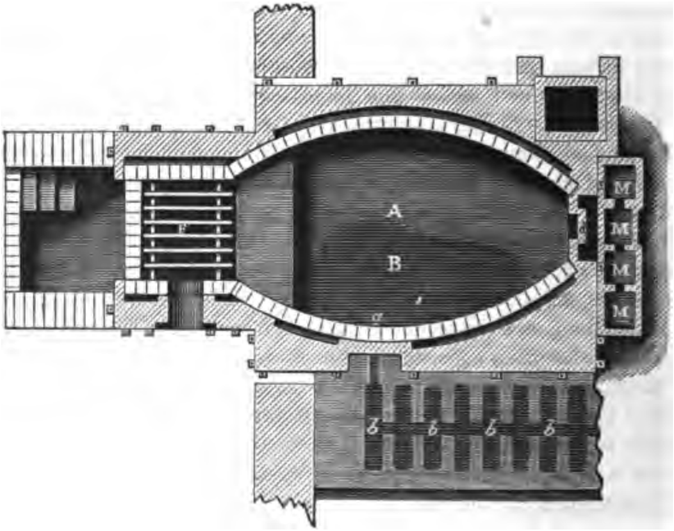

Fig. 122.—Melting-Furnace; horizontal section.

tion is conducted as before; this is repeated until, at the expiration of twenty-four hours, the cavity of the furnace has become full of regulus, when the tap-hole, *a*, is opened, and the regulus or coarse-metal runs off into the sand-moulds, *b*, and is afterwards crushed, either between rolls or by a stone-breaker, previous to calcination. The slags from this furnace should not, on an average, contain much above $\frac{1}{2}$th per cent. of copper. The resulting slags, chiefly consisting of silicates of iron, contain numerous disseminated fragments of quartz, which give to the whole a mottled appearance and somewhat pasty consistency. It is important that these slags should have a proper degree of fluidity, since if too stiff, they are liable to retain shots of regulus, and if too thin, the workmen find it difficult to skim them from the top of the charge without drawing out at the same time a portion of the matte.

Any regulus which may be thus accidentally drawn out of the furnace collects, for the most part, in the bottoms of the sand-moulds in which the slags are received, and is afterwards carefully removed. The slags, when sufficiently cold, are broken and subjected to careful examination; those portions which contain a sufficient amount of regulus are preserved for re-melting with the roasted ores, while the remainder is rejected as useless.

III. *Calcination of Crushed Coarse-Metal.*—The furnace employed for calcining the coarse-metal usually resembles in all respects that used for crude ores in the first operation. The object of this process is the elimination of a further portion of the sulphur and the oxidation of iron, which is the more readily effected on account of the removal of the earthy and siliceous matters present in the raw ores. The charge varies from 3½ to 4 tons, and the calcination is usually complete in twenty-four hours. When the coarse-metal is first introduced into the furnace, the temperature is for some time carefully regulated, the heat being afterwards cautiously increased, until, at the expiration of about fourteen hours, bright redness has been attained. This temperature is maintained until the charge has been in the furnace twenty-four hours, care being taken to stir and turn it over from time to time. The plates covering the holes, *e*, fig. 120, are now removed, and the charge is scraped through them into the chambers, C, beneath the furnace.

IV. *Fusion of Calcined Coarse-Metal with Ores belonging to Class* 3, *and Slags from Operations V. and VI.*—The object of this fusion, which occupies from five to six hours, is to eliminate, in the form of silicate, a further portion of iron, and to produce a regulus richer in copper than is coarse-metal; the products are *white-metal* or *pimple-metal*, very rich in copper, and *metal-slags* melted in operation II. The fusion is effected in a furnace so similar to that employed for operation II. as to require no special description. At this period are introduced rich foreign oxides and carbonates, belonging to Class 3, containing but little iron. The charge generally weighs from 50 to 52 cwts., and is constituted in accordance with the nature of the ores which the smelter may have at his disposal.

It is often made up nearly as follows:—30 cwts. calcined coarse-metal, 16 cwts. of rich carbonates and oxides, and 5 cwts. of roaster- and refinery-slags from operations V. and VI. In addition to these, copper-scale and furnace-bottoms are occasionally added in certain proportions. In all cases the charge should be so constituted that nearly the whole of the sulphide of iron present may become decomposed at the expense of oxide of copper, and that the oxide of iron so formed may, in the form of silicate, pass off in the slags. Towards the close of the operation the charge is well stirred, and shortly afterwards the slag is skimmed off and drawn out of the furnace through the door, *d*, below which sand-moulds are prepared for its reception. The regulus is finally tapped off into sand-moulds beneath the tap-hole, which is on the side of the furnace, and should be in the state of *white-metal* or *pimple-metal*, containing from 76 to 79 per cent. of copper. This sulphide is very nearly represented by the formula $Cu_2S$, although it always retains small quantities of iron.

V. *Roasting White-Metal or Pimple-Metal.*—This operation is carried on in a reverberatory furnace, similar to the ordinary melting-furnace, provided with air-ways in the bridge-end, and with a lateral door through which the pigs of regulus from operation IV. are introduced. The products are *blister-copper*, containing above 99 per cent. of copper, and *roaster-slag.* The pigs of regulus are placed on the blade of a long paddle, and each is transported to its proper place on the hearth of the furnace, of which the temperature becomes considerably reduced during the introduction of the charge. The weight of the charge is ordinarily from 4 to 5 tons, and the heat is so regulated that the pigs may be completely melted at the expiration of from six to eight hours. During this time air is allowed to circulate freely through the furnace, and sulphurous anhydride is abundantly evolved. The slag formed on the surface of the melted regulus is frequently skimmed off during the progress of the operation; the first time immediately after the complete fusion of the charge, and the last shortly before tapping. A peculiar frizzling sound is emitted from the bath of fused regulus, which is maintained in a state of constant ebullition. After the contents of the furnace have been for a considerable time in a state of fusion, the temperature is sufficiently lowered to cause the surface of the regulus to become pasty, and it is then thrown up into crater-like elevations, produced by the escaping gases. When the temperature begins to fall too low, the door is closed and the mass again brought to a fusing heat, at which it is maintained for some hours, during which time sulphurous anhydride continues to be freely evolved.

Before the close of the operation, which may last from twenty-four to forty-eight hours, the openings admitting air into the furnace are shut, and the slags resulting from the combination of silica, derived from the hearth and from the sand adhering to the pigs of regulus, with oxides of iron and copper, are skimmed off the surface, and the blister-copper is tapped into sand-moulds. The slags resulting from this operation are added to the charge in operation IV.

VI. *Refining and Toughening.*—The furnace employed for the operation of refining is similar to the ordinary roasting-furnace, excepting that its bottom inclines in all directions towards a point near the end door, next the chimney. There is also another door at the side, but there is neither a hole in the roof for charging nor a tap-hole. The products obtained are *marketable copper* and *refinery-slag;* the latter ultimately forming part of the charge in operation IV. The charge of the refinery consists of about 10 tons of blister-copper, which is introduced through the side door, and is piled in the form of a hollow heap, extending to the arch of the furnace; the cakes being so arranged as to allow a sufficient space for the free circulation of air between them. The complete fusion of the charge is usually effected in about four hours, when the slags are removed by skimming, and the fused metal is exposed during from fourteen to sixteen hours to the oxidizing influences of the air passing through the furnace. The charge is from time to time rabbled and the slag skimmed off, and at the expiration of the time specified the charge should be in the state of *dry copper.* In order to see whether the process of oxidation has been sufficiently prolonged, the refiner takes out a sample in a small iron ladle, and from the fracture of this sample he is enabled to judge of the progress of the operation.

Copper in this *dry* state contains a certain proportion of oxygen in combination, and in order to eliminate this it is subjected to the process of *toughening.* When the charge is found to be sufficiently saturated with cuprous oxide, the slag is skimmed off, and two or three shovelsful of anthracite or charcoal are thrown on the hearth and spread over the surface of the liquid metal. This covering of carbon tends to the reduction of the oxide of copper formed on the surface of the metal, and after a short interval, during which the charcoal is allowed to act alone, a long pole of green wood is plunged into the fused copper. Under the influence of the elevated temperature to which the wood is thus exposed, large quantities of reducing gases are evolved by its decomposition, attended with strong ebullition of the metal. The reduction of the oxide is thus determined with much greater rapidity than by the action of charcoal or anthracite alone; the latter nevertheless not only assists in the removal of oxygen, but also prevents the absorption of a further amount when the surface of the liquid metal is in a state of rest.

When the fused copper has, by this means, been kept in a state of ebullition during some time, usually an hour or more, the refiner takes a sample from the furnace by inserting into it a small ladle-shaped mould about 1½ inch in diameter and ½ inch in depth. The sample thus obtained, which is thicker in the centre than at the circumference, is tested with regard to malleability by flattening on an anvil, and after being partially cut through with a cold-chisel is fixed between the jaws of a vice, and bent backwards and forwards until broken.

As soon as the charge is found, from these trials and from the colour of the copper, to have reached *tough-pitch*, the pole is taken out and the charcoal pushed back from the opening, through which the metal is removed by iron ladles and transferred to copper moulds. These moulds are cast in a press from the ordinary charges of copper, and when cracked or otherwise rendered unfit for further use, are thrown into the refinery and re-melted.

The moulds, *a*, fig. 123, when in use are fixed by a dove-tailed bottom to iron *tipplers*, *b*, arranged along one edge of an iron trough, *c*, and when the copper laded into the mould has completely set, the tippler with its attached mould is turned over with a smart shock, so that the ingot is turned out into water. On turning it back into its original position, any water which may have splashed into it is rapidly dried up by the heat of the mould, which is then ready for re-filling.

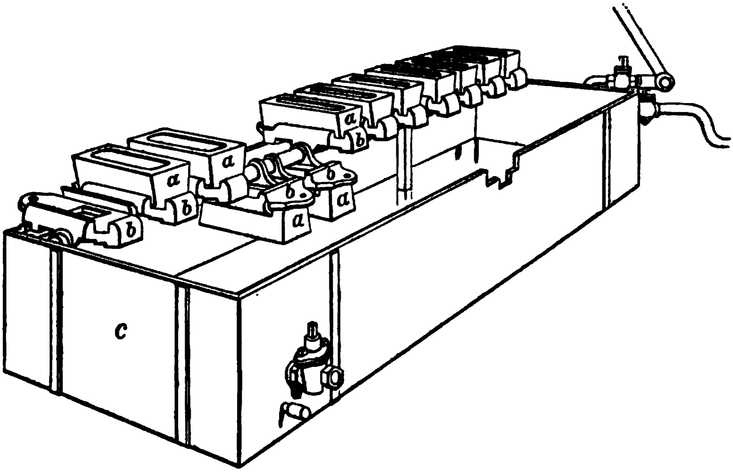

Fig. 123.—Ingot-Moulds.

There are usually three troughs, each carrying twelve moulds, placed at a convenient distance from the end door of the refinery, and during the lading of a charge of copper a current of cold water is kept continuously flowing through the troughs.

When the copper is found to be in a proper state for removal from the furnace, it is necessary that this should be effected with the least possible delay, since it would otherwise be liable to again become somewhat dry through the absorption of oxygen. Should this occur, poling must for a short time again be resorted to, and if the metal becomes *over-poled*, this defect is readily obviated by a short exposure to the oxidising influences of the air. The surface of an ingot at tough-pitch is approximately level, whereas under-poled copper sinks into a longitudinal furrow, whilst over-poled copper rises into a ridge. When copper is intended for rolling, a few pounds of lead are sometimes added and well mixed with the charge immediately before it is laded into moulds, which in this case are of cast-iron. These are first washed on the inside with tan-ashes and water, and thoroughly dried. A false bottom of copper is then formed by pouring a couple of ladlesful of the charge into each pot or mould; and when this has set, the requisite number of ladles of copper to form a cake of the desired thickness are laded on to the top of it. When this cake has sufficiently cooled, it forms the bottom on which the next cake is cast; each mould, when filled, contains from four to six cakes of metal, according to thickness. When cold, the mass is turned out from the pot, the cakes separated, and their edges trimmed; the copper used to form the false bottom in the first instance requires to be re-melted, as, being full of blow-holes, it is not sufficiently sound for rolling.

When very large cakes are required, the moulds are often made by clamping together L-shaped pieces of cast-iron on a cast-iron plate.

*Use of Phosphorus in Refining.*—The addition of phosphorus to copper during that part of the operation of refining known as "poling" appears to increase the density of the refined metal, and at the same time to so assist the deoxidation as to dispense, to a considerable extent, with the use of the pole. It is not, however, desirable to effect deoxidation entirely by the use of phosphorus, but rather by its agency to bring the metal nearly to the tough-pitch, and then to finish the operation by the use of the pole. The phosphorus is added in the form of phosphide of copper, containing about 7 per cent. of phosphorus, which is prepared by pouring melted copper upon phosphorus in the bottom of an iron crucible, which is protected by a lining consisting of a mixture of loam and powdered coke. The cover of the crucible is fitted with a funnel, and an aperture is left for the escape of gases, while the metal falls upon a perforated iron diaphragm, coated like the inside of the crucible itself, placed midway between the bottom and the cover. The metal thus falls in the form of rain, and is therefore in a suitable condition for absorbing the escaping vapour of phosphorus.

The proportion of phosphorus required necessarily depends upon the dryness of the copper to which it is added; but at Chatham Dockyard, where the process has for some time been successfully employed, 1 cwt. of copper phosphide is introduced into a 5-ton charge, and with this quantity the copper is rarely over-poled. This represents about 0·07 per cent. of phosphorus, and about one-half of that quantity is permanently retained by the copper.

*Process of making "Best-Selected" Copper.*—Dr. Percy, who derived his information from Mr. Keates, of the firm of Newton, Keates, & Co., makes the following observations on the subject of best-selected copper:[1] "The introduction of the manufacture of brass, on a large scale, into this country does not date much farther back than the year 1680, and the manufacturers were not long in discovering that copper taken indiscriminately, as it occurred in the market, frequently produced brass quite unfit for manufacturing into battery, sheets, and wire, and they rightly attributed this to its impurity. The English copper generally in use at the beginning of the eighteenth century was derived from Cornish ores, which were then, to a greater extent than at present, mixed with tin; and it is most creditable to the sagacity and practical skill of the smelters of that day that they devised a mode of remedying the evil which, in effect, has not been improved upon by their successors."

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[1] 'Metallurgy: Fuel, Copper, &c.,' p. 329.

The details of the processes by which copper of this quality is prepared from ordinary ores vary considerably in different works, although the principle involved is in all cases the same. Advantage is taken of the circumstance that when copper ores contaminated by the presence of other metals, such as arsenic, tin, lead, &c., are reduced to a state of regulus, and afterwards so roasted as not to contain a sufficient amount of sulphur to convert the whole of the metals into regulus, a certain portion of the copper will, during the subsequent fusion, be liberated in the metallic form. The copper so set free falls to the bottom of the moulds in which the contents of the furnace are tapped, and retains a very large proportion of the impurities by which the quality of the product would be impaired. The material usually operated on is pimple-metal, which is melted down and roasted during a certain time, in accordance with the judgment of the furnace-man. When ready, the contents of the furnace are tapped into a series of sand-moulds, joined together by gutters in the upper parts of the partition walls which separate them from one another. The mixture of regulus and copper alloy containing the impurities to be separated is tapped into the first of these moulds, and when this becomes full, the melted matter flows over into the second, and so on until the whole of the charge has been run out. The total number of moulds may be from sixteen to eighteen, according to the weight of the charge, and the size of the sand-beds prepared to receive it, and from six to eight pigs of impure copper will be found in the bottoms of those nearest the tap-hole. The *regule* is removed, by a hammer, from the tops of each of these as soon as it has sufficiently cooled to admit of being conveniently handled. About one-fourth of the total amount of copper present may be reduced in this method of making *best-selected* copper, the regule obtained being again, if necessary, subjected to similar treatment; a further production of impure copper is the result of the second fusion. The total amount of copper thus abstracted as impure may be nearly one-half the quantity contained in the material originally charged into the furnace. The *bottoms* obtained as the result of these fusions may, according to circumstances, either be refined, cast into rectangular plates, and sold as *tile-copper*, or, if found advantageous, it may be made into *cake-copper*.

MODIFICATIONS OF THE WELSH METHOD OF COPPER-SMELTING.—As before stated, the general routine of the different processes employed for the metallurgical treatment of copper ores by the Welsh process is not only varied in accordance with the quality of the copper it is intended to produce, but also to meet the circumstances of the varying nature of the supply of cupriferous materials available.

The large quantities of copper-precipitate now supplied by the numerous extraction-works, added to the amount of Chilian and American regulus which is imported into this country, have resulted in the introduction, in the copper-works in the neighbourhood of St. Helens, where a large portion of this precipitate is worked up, of certain modifications of the ordinary processes.

The following are the various operations usually followed in the copper-works at St. Helens:—

I. Calcination of ores.
II. Melting for coarse-metal.
III. Melting for white-metal.
IV. Tapping close-regulus.
V. Running into blister-copper.
VI. Refining and toughening.

I. *Calcination of Ores.*—A charge of from 5 to 6 tons of sulphurous ores and regulus is introduced into a Siemens furnace heated with gas, supplied from a generator. The ore is spread over the bed, and the flame circulates around the brickwork of a chamber like an oven or muffle, which is in communication with sulphuric-acid chambers. The charge is stirred every two hours through the doors, and is kept at a red heat during about ninety-six hours, when it is drawn with rakes, and is usually found to contain from 6 to 7 per cent. of sulphur.

II. *Melting for Coarse-Metal.*—This is carried on in an ore-furnace, in which the *sharp-slags* obtained from the skimming of No. III. process are worked. This is the usual reverberatory furnace, and is heated with slack. A charge of about 36 cwts. is introduced and kept at a strong heat for about six hours. When the door is removed, and the charge is found to be in a liquid state, the coarse-metal falls to the bottom and the fusible slag floats above it. The furnace-man then skims off through the front door the slag, which should be free from copper, or ought at any rate not to contain more than ¼th of 1 per cent.

Once every twenty-four hours the coarse-metal is tapped into sand-beds at the side of the furnace, and should not contain above 33 per cent. of copper. The charge consists of poor calcined ores and a small quantity of poor raw ores, with the sharp-slags from No. III.

III. *Melting for White-Metal.*—A charge of about 40 cwts. of a mixture of coarse-metal from the preceding operation, Chili regulus, calcined rich sulphides, and raw carbonates, is placed in a furnace similar to the last. The door is closed, and the heat increased for about four hours, when the charge will be found to be in a liquid state. The furnace-man skims off the slags floating on the surface, until he comes to the metal, which he can readily distinguish. The slags from this operation are termed "sharp-slags," and are those melted in operation II. for coarse-metal. The metal is tapped at the side, exactly as in operation II., and contains about 66 per cent. of copper.

IV. *Tapping Close-Regulus.*—This process is termed "selecting," as the metal is divided into two qualities, one for making best copper and the other for common.

About 40 cwts. of a mixture of coarse-metal, calcined regulus, calcined rich sulphide ores, and copper-precipitate obtained from the various extraction-works, is placed in a furnace of the same construction as the last, where it remains until the charge becomes perfectly liquid. It is then tapped at the pitch called *close-regulus*, when it is found that, in combination with reduced copper, the impurities fall through the regulus to the bottom of the moulds. When cold, the workman separates the upper portions from the lower by means of a hammer.

The upper portion consists of sulphides of copper; the lower of metallic copper which contains other metals, such as tin, antimony, arsenic, lead, &c.

V. *Running for Blister-Copper.*—This is effected in a furnace called a "roaster," in which three kinds of material are separately treated, according to the quality of the copper required, namely, *white-metal*, from No. III. process; *close-regulus*, from No. IV.; and *bottoms* separated from it.

One method of treatment answers for all:—About 7 tons of white-metal are placed in the roaster, the temperature of which is raised, and the metal reduced, at first slowly, to a liquid state, while a current of air is allowed to pass freely through the furnace. When it has been working about forty-eight hours, the evolution of sulphur will be observed to have ceased, and the sulphides will be converted into blister-copper, which is run into sand-beds. The same is done in converting close-regulus and bottoms into "blister;" in the first case about thirty hours are required, in the latter only eighteen, owing to the bottoms containing much less sulphur than the close-regulus.

VI. *Refining and Toughening.*—A charge of about 10 tons of blister-copper is put into the refining-furnace, which is of the same construction as the other copper furnaces, excepting that the bottom slopes down towards the front door, where there is a cavity to enable the men to dip their ladles when lading out the copper. The charge is roasted for about four hours at a gentle heat, with a current of air, in order to liberate any sulphur that may remain after the last operation.

The heat is now raised until the metal becomes liquefied, when it is skimmed to free it from slag. After having been alternately rabbled and skimmed for about six hours, it is brought to the pitch of *dry copper*. It is then agitated or poled with large poles, so as to remove oxygen, by which means the copper is brought to *tough-pitch*. Samples are removed by small ladles, hammered and broken in the vice; when the fracture and colour of the metal indicate that it is in a fit condition, it is at once laded into moulds.

In Chili, where the ores consist of a mixture of sulphides, oxides, carbonates, silicates, and oxychlorides of copper, smelting is conducted in reverberatory furnaces, with coal as fuel, and comprises three principal operations only, namely, fusion for regulus, calcining for spongy regulus, and roasting for blister-copper. The copper arrives in this country in *bars*, usually weighing about 1¾ cwt. each, and, according to its state of purity, it is either at once refined or subjected to a preliminary roasting.

In this country "best-selected" copper usually fetches about £1 per ton above the price of ordinary tough-ingot.

The usual forms in which copper is sent into the market are as follow:—

| | | | | | | | | | cwts. | qrs. | lbs. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Cakes | . | 19 in. | × | 12½ in. | × | 1⅞ in.; | weight | | 1 | 1 | 0 |
| Tiles | . | 19 „ | × | 12½ „ | × | ½ „ | „ | | 0 | 1 | 13 |
| Ingots | . | 11 „ | × | 3½ „ | × | 1½ „ | „ | | 14 to 16 lbs. | | |

## TREATMENT OF CUPRIFEROUS SCHISTS IN THE MANSFELD DISTRICT, PRUSSIAN SAXONY.

The ore treated is the well-known *Kupferschiefer* of the Germans, a bituminous schist or shale, which occurs below the *Zechstein*, a formation of Permian age. The thickness of the copper-bearing shale is seldom above 18 inches, and of this a few inches only will usually repay the expenses of smelting. Mining, which is sometimes carried on at a depth of eighty fathoms from the surface, is particularly laborious in this district, since the thinness of the deposit renders it necessary for the miner to conduct all his operations while lying on his side. Smelting has long been carried on in the neighbourhood of Mansfeld. Agricola, who wrote about the middle of the sixteenth century, minutely describes the way in which the ores were burned in heaps in the vicinity of Eisleben, as a preliminary to fusion. The existing smelting-works are situated near the towns of Mansfeld, Eisleben, and Sangerhausen. The various mines and smelting-works, which were before 1852 possessed by separate companies, were at that date united under a single direction, of which the chief office is in Eisleben; the result being the establishment of an admirable system of management, which has secured large and continuous profits to the proprietary.

The following analyses, made by Scheerer in 1879, give the composition of unburnt Kupferschiefer from four different localities:[1]—

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[1] 'Mansfelder Kupferschieferbauende Gewerkschaft.' Eisleben, 1881.

| | Otto Shaft. Right Side. | Otto Shaft. Left Side. | Ernst Shaft. Left Side. | Glückhilf Shaft. Left Side. |
|---|---|---|---|---|
| $SiO_2$ | 38·42 | 32·87 | 33·15 | 29·22 |
| $Al_2O_3$ | 15·93 | 11·28 | 12·90 | 11·76 |
| CaO | 10·93 | 14·31 | 14·39 | 12·66 |
| MgO | 3·53 | 4·53 | 2·32 | 2·25 |
| $CO_2$ | 7·02 | 13·51 | 10·47 | 9·43 |
| Fe | 1·81 | 0·85 | 3·31 | 2·97 |
| Cu | 2·01 | 2·93 | 2·90 | 2·88 |
| Ag | 0·015 | 0·010 | 0·016 | 0·021 |
| S | 3·18 | 3·96 | 2·15 | 4·97 |
| Bitumen, &c. | 14·63 | 14·07 | 9·89 | 17·21 |

Zinc, lead, manganese, nickel, and cobalt were not quantitively estimated.

The Kupferschiefer of the Mansfeld district contains on an average somewhat less than 3 per cent. of copper, with about 10 lbs. of silver to the ton of copper. The *Sanderz* of the Sangerhausen district sometimes contains as much as 5 per cent. of copper when the ore is copper pyrites; but when it consists of erubescite, it occasionally reaches 10 per cent.; in the latter case the proportion of silver is reduced to one-half.

The fuel employed is principally coal and coke, a considerable portion of which reaches the works from England *viâ* Hamburg; another portion is, however, supplied by the German coal-fields, and gas-coke is collected from the neighbouring towns and cities where large quantities of gas are consumed. Formerly wood and charcoal were exclusively made use of in the smelting-works; but, although the company still possesses a large extent of woodland, it is now found more profitable to sell the timber and charcoal, and to obtain from a distance supplies of fuel better suited for the work. Brushwood answers as fuel for muffle-furnaces and for calcining; the brown coal of the neighbourhood, when mixed with coal of a superior quality, is also employed for reverberatory furnaces. In addition to slags, which, for the sake of freeing them from copper, are passed through the furnace with the various charges, fluor-spar is sometimes employed. It is chiefly used with highly siliceous materials, and is found abundantly in the neighbourhood of some of the works.

As at present conducted, the method employed for the treatment of cupriferous schists in the Mansfeld district comprehends the following operations:—

I. Burning the schist in heaps for the purpose of removing a portion of the sulphur, together with water and bitumen, and to reduce the material to a mechanical condition suitable for smelting.

II. Smelting the burnt schist with slags and fluor-spar in blast-furnaces; products, *coarse-metal* or *Rohstein* and slags; the latter being sometimes moulded into blocks for building purposes.

III. Burning the coarse-metal in kilns for the purpose of eliminating sulphur, oxidizing the iron, and producing sulphuric acid.

IV. Concentration of the copper in the roasted coarse-metal by fusion in reverberatory furnaces; products, *fine-metal* or *Spurstein*, containing 74 to 75 per cent. of copper, with silver, and *rich slag* sent back to operation II.

V. Grinding the fine-metal.

VI. Roasting the ground fine-metal; the chief portion of the copper is thus transformed in *cupric oxide*, while the silver is converted into a soluble *sulphate of silver.*

VII. Dissolving out the sulphate of silver with warm water, and precipitating *cement-silver* from the solution by means of metallic copper.

VIII. Mixing the residues from this lixiviation with coal-dust, and melting and refining in a reverberatory furnace; products, *refined copper* or *Kupferraffinad* and *slags.*

IX. Treatment of slags, resulting from operation VIII., for copper of second quality.

The following description of the various processes employed in the Mansfeld district for the treatment of Kupferschiefer will render intelligible the series of manipulations to which it is subjected for the extraction of copper and silver.

I. *Burning the Schist.*—This has for its object the combustion or volatilization of a large proportion of the bitumen, as well as the expulsion of water, &c.; a portion of the sulphur is also at the same time eliminated, but care must be taken to retain a sufficient quantity to form good coarse-metal with the copper and a portion of the iron. This operation, which partakes more of the nature of burning than of calcination, is accomplished in large heaps constructed in the vicinity of the smelting-furnaces, but generally at a higher level.

In order to construct a heap of this kind, a number of faggots of dry brushwood are laid side by side on the pavement of the roasting-yard in such a way as to mark out the intended boundary of the mound, thus enclosing a space of from 200 to 300 feet in length, and 30 to 40 feet in width. This area is traversed, longitudinally, by a line of faggots arranged along its centre, across which two or three rows of similar faggots are placed at right angles. Where these rows cross one another a small pile of faggots is erected. The schist is piled loosely upon the faggots until a rectangular heap, from 7 to 10 feet in height, has been formed; fire is now applied to the wood on one side, and the flames gradually spread, through the channels filled with faggots, to the heaps of fuel at their intersection, which act as chimneys to the mass. Schist made into heaps shortly after its extraction from the mine is found to burn more readily than when put together dry; this arises from the fact, that as soon as the water is expelled, the layers of shale open, leaving interstices by which the mass is rendered permeable to the air, whereas, if previously dried, it crumbles, and a compact heap is the result. The best conditions are obtained when the blocks are put together in a frozen state during the winter, and gradually thaw after the completion of the heap. When the schist has once become well ignited it goes on burning slowly until the whole of the bitumen has been consumed. This occupies a greater or less time, in accordance with the state of the weather and the size of the heap, but the smaller heaps generally require from eight to ten weeks, and the larger from three to four months. It sometimes happens that during very strong winds so much heat is developed as to cause the ore to melt and run together into masses. This not only results in considerable additional expense, as it makes the heaps difficult to break up, but also so completely drives off the sulphur that the roasted material requires to be smelted with raw ores containing sulphides. In order to avoid this inconvenience, it is usual to erect screens of rough boarding in the direction of the prevailing winds, or to cover the exposed portions of the heap with a layer of finely divided and closely packed ore. The following analyses, made by Grunow in 1850, give the composition of burnt Kupferschiefer from four different localities:[1]—

| | Lower District. | Schaafbreit District. | Cuxberg District. | Sangerhausen District. |
|---|---|---|---|---|
| $SiO_2$ . . . . . . | 49·00 | 50·00 | 49·20 | 52·92 |
| $Al_2O_3$ . . . . . . | 15·31 | 18·00 | 16·00 | 15·67 |
| $Fe_2O_3$ . . . . . . | 8·00 | 8·00 | 8·00 | 8·00 |
| CaO . . . . . . | 18·10 | 13·20 | 15·60 | 9·99 |
| MgO . . . . . . | 4·14 | 3·30 | 4·50 | 4·65 |

Ten pounds of wood are, on an average, consumed for each ton of schist burnt; the reduction in bulk which takes place during the operation is about 10 per cent., and the loss of weight 16 per cent.

II. *Smelting Burnt Ore with Slags for the Production of Coarse-Metal or Rohstein, &c.*—The roasted ore is taken from the pile in which it is burnt directly to the smelting-furnace, where it is fused with a mixture of slags and fluor-spar, the products obtained being coarse-metal and poor slags. This fusion always takes place in blast-furnaces, of which the form and dimensions vary considerably. The older furnaces were rectangular, and from 15 to 20 feet in height; the newer ones, on the contrary, are circular, and have a total height of above 30 feet. Figs. 124 and 125 represent one of the rectangular furnaces formerly employed at Mansfeld for the fusion of roasted schist. The first is a front elevation, and the second a vertical section through the axis of one of the tuyers; the lining in the vicinity of the hearth was constructed of a sandstone found in the neighbourhood; the upper portion, A, was lined with fire-brick, and the outer walls were built with ordinary masonry.

The blast was supplied through tuyers, *t*, placed either in the back of the hearth or in two opposite lateral faces of the furnace, and at the same height from the bottom. On a level with the floor were two aper-

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[1] The metallic sulphides were not estimated.

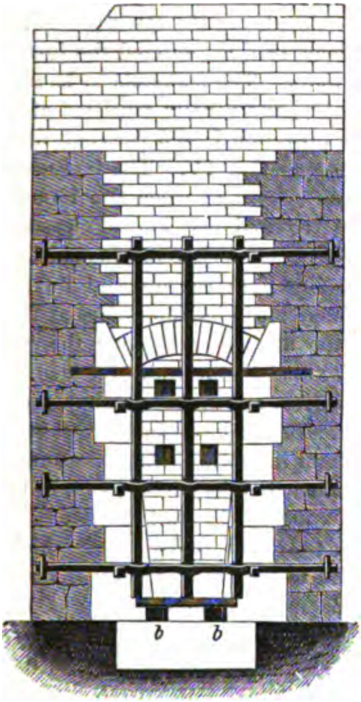

Fig. 124.—Rectangular Furnace, Mansfeld; elevation.

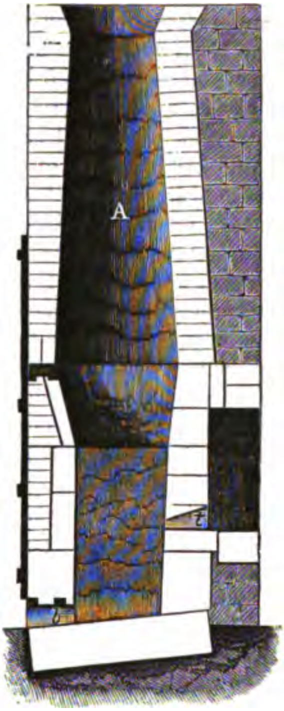

Fig. 125.—Rectangular Furnace, Mansfeld; vertical section.

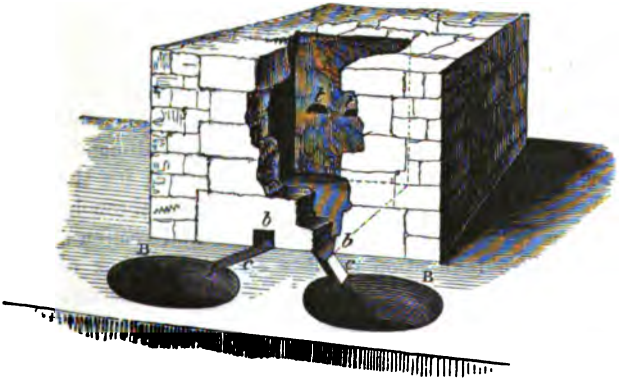

Fig. 126.—Rectangular Furnace, Mansfeld; interior of hearth.

tures, *b*, fig. 126, communicating, by the channels, *c*, with two external basins, B, each about 30 inches in diameter and 14 inches in depth, hollowed in a bed consisting of clay and coke-dust. The slags and mattes flowing constantly out of the furnace were received into one or other of these basins, and when one had become full the aperture by which it communicated with the hearth was closed, and the other opened.

The blast, which was heated to about 185° C., entered the furnace at a pressure of from 7 to 8 inches of water. The materials to be charged were placed on a platform near the top, and usually consisted of about 86·5 per cent. of roasted schist from operation I., 6·5 per cent. of fluor-spar, and 7·0 per cent. of slags from operation IV. The fuel used was either English or Westphalian coke, or gas-coke; when the former was employed, 12 to 14 cubic feet were required per ton of roasted ore smelted, but in the latter case 16 to 18 cubic feet were consumed. The fuel was introduced in layers, alternately with the ore and flux, and a fresh charge was added as soon as flame made its appearance at the top. In this way the complete fusion of the mass was effected; the gangue, uniting with a certain proportion of oxide of iron, formed a fusible slag, while the copper, in combination with iron, silver, and sulphur, yielded a liquid regulus or coarse-metal. The slags and regulus flowed together into the basins, B, where the latter, from its greater density, accumulated at the bottom, while the lighter slags floated on its surface, and, in proportion as the basin filled, were dragged aside by the workmen.

The basins, B, being used alternately, the coarse-metal which had collected in one of them was allowed to cool while the other was being filled, and, when sufficiently set, was removed in the form of circular plates, which were lifted from the surface of the still liquid portion remaining in the bottom of the cavity. This was done by an iron eye inserted in the regulus while in a liquid state.

As soon as a plate had been separated it was broken with sledges, any adhering pieces of slag being picked out and returned to the furnace in which the fusion was effected. The coarse-metal obtained amounted to about 10 per cent. of the weight of the burnt schist smelted; it contained from 30 to 40 per cent. of copper and about ⅛th of 1 per cent. of silver together with iron, cobalt, nickel, zinc, and sulphur.

The small rectangular blast-furnaces have at Mansfeld been gradually replaced by circular furnaces of much larger dimensions; these are blown by six tuyers and provided with apparatus for the collection of the waste gases. Fig. 127 is a vertical section through the centre of one of these furnaces, and fig. 128 a horizontal section at the level of the tuyers. The foundation consists of a solid block of masonry, A, provided with proper channels for the escape of moisture. The furnace is supported on eight short cast-iron pillars, *a*, carrying an iron ring, *b*, and is lined with fire-brick. The blast, which is heated to a temperature of 280° C., enters the furnace by six water-tuyers, *c*, under a pressure of about 2 lbs. per square inch, while the throat, which is closed by the cup-and-cone arrangement, *d*, admits of the waste gases being collected by the openings, *e*, and the wrought-iron pipes, *f*. The charges are introduced in the usual way; the slags flow off constantly at *g*, and the coarse-metal is from time to time tapped off at *h*, on the opposite side of the furnace. This formerly ran through the iron gutter, *i*, into the cross-spout, *k*, fig. 128, with serrated ends, from which it fell into

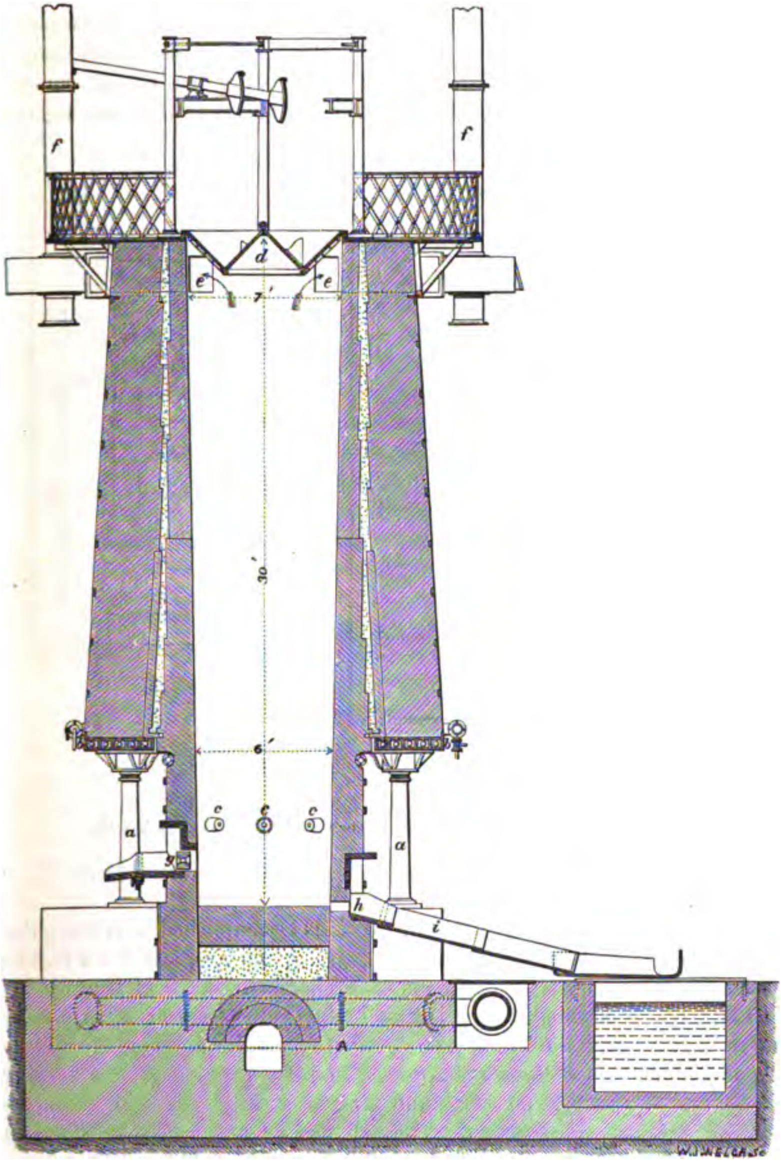

Fig. 127.—Six-tuyer Furnace, Mansfeld; vertical section.

a cistern of water, where it was granulated; but it is now collected in moulds.

A furnace with two tuyers, and blast heated to 100° C., will smelt from 7 to 8 *fuders* (21 to 24 tons) of burnt schist in the course of twenty-four hours; a furnace with three tuyers will smelt 12 fuders in the same time. With a furnace blown by four tuyers, 17 fuders of ore can be smelted in twenty-four hours, while a large furnace with six tuyers smelts from 40 to 45 fuders of ore, with an expenditure of from 12 to 14 cwts. of coke per fuder.

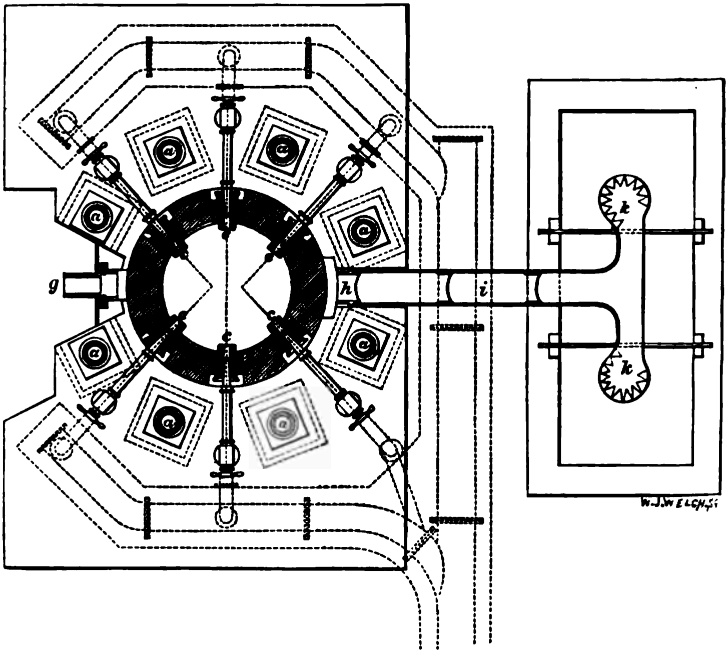

Fig. 128.—Six-tuyer Furnace, Mansfeld; horizontal section through hearth.

III. *Roasting the Coarse-Metal.*—This is accomplished in rectangular kilns, each 10 feet in height, 5 feet square at the top, and 5 × 4 feet at the bottom, provided with cup-and-cone chargers. Each of these holds 10 tons of coarse-metal, which is first broken into pieces of a little more than a cubic inch, and then introduced through the charging apparatus in the roof. These kilns are arranged in rows, forming groups varying from ten to twenty, and communicating with sulphuric-acid chambers of the ordinary construction. The calcined or burnt matter is drawn every twelve hours, each kiln yielding from 1¼ to 1½ ton of burnt material every day of twenty-four hours.

In order to prevent the clotting and fusing-together of the unroasted coarse-metal, it is mixed with nearly an equal weight of the same material which has been already roasted; the proportions being 100 parts of unroasted coarse-metal to 85 parts of the roasted metal. A slight increase of weight, due to the oxidation of iron, takes place during the operation, as 1 part of unburnt coarse-metal is, on an average, represented by 1·042 of burnt residue. The burning is going on most satisfactorily when the heated metal slightly sticks together at a height of about 3 feet above the level of the bottom; thus forming a scaffold which enables the burnt ore to be withdrawn, before breaking down the platform and allowing the material in the upper parts of the kiln to descend.

IV. *Melting for Fine-Metal or Spurstein.*—The furnace employed for this operation closely resembles that used in this country for melting for coarse-metal. Until within the last twenty-five years the roasted coarse-metal was fused in a small blast-furnace, but this has been superseded by the Welsh reverberatory furnace introduced from Swansea at about the date stated.

The charge usually consists of 50 to 56 cwts. of roasted coarse-metal, and 4 to 7 cwts. of siliceous ores, or 2½ to 3 cwts. of sand. This mixture is charged into the furnace through a hopper in the usual way. When first charged, the furnace is closed and fired sharply for three and a half hours, at the expiration of which time the first rabbling takes place; this rabbling is repeated twice, and when, at the expiration of about eight hours, the charge has become reduced to a perfectly liquid condition, and no lumps are found upon the bottom, tapping takes place. The tap-hole is opened and the fine-metal is run into a series of conical cast-iron moulds, each capable of holding from 4 to 6 cwts. of regulus, so arranged that the second is slightly below the first, the third below the second, and so on. Only the purest and richest materials produce very liquid slags; when the matte contains zinc the resulting slags are infusible and pasty. Spurstein should contain from 74 to 75 per cent. of copper, and when it either falls below 72 per cent. of copper, or contains that metal in the metallic state, it is not forwarded to the silver-extraction works. The presence of metallic copper in the fine-metal is obviated by the addition of a little unroasted Rohstein to the charge. The slags from this furnace contain copper, and are smelted in operation II., with roasted schist and a little fluor-spar. The fuel employed is a mixture of brown coal and English bituminous coal, in the proportion of two of the former to one of the latter, burnt on a step grate.

V. *Grinding the Fine-Metal.*—When sufficiently cold, the Spurstein is removed from the cast-iron cones, freed from any adhering slag, and reduced to the state of an impalpable powder. This is effected in ball-mills, each capable of grinding 20 tons of matte in the course of twenty-four hours.

VI. *Roasting the Ground Fine-Metal.*—The concentrated and finely ground sulphide, which contains about 74 per cent. of copper and about 0·43 per cent. of silver in the metallic copper, is subjected to a process of careful roasting, by which the copper is, for the most part, converted into an insoluble oxide, while the silver is transformed into a readily soluble sulphate of that metal. This is effected in reverberatory furnaces, with precautions which will be stated when describing Ziervogel's Silver Process.

VII. *Dissolving out Sulphate of Silver and Precipitating Cement-Silver by Metallic Copper.*—This operation is conducted in a series of tubs, in some of which the solution of sulphate of silver is effected, while in others the precipitation of the dissolved silver is determined by the introduction of metallic copper, both in the form of bars and in a granulated state. The method of conducting this operation will be described when treating of the metallurgy of silver.

VIII. *Fusion and Refining of Desilverized Residues.*—The residues retained in the tubs in which the lixiviation for sulphate of silver has been conducted, contain about 75 per cent. of copper, chiefly as oxide, and have been freed from silver to within 0·018 per cent. This material is reduced to the metallic state, and is subsequently refined in a furnace, which resembles the ordinary English refinery. Ten tons of the washed residue from the silver-extraction process, after being allowed to drain freely, are dried upon beds warmed by the waste heat from the flues, mixed with 10 per cent. of coal-dust, and charged into the furnace.

At the expiration of from nine to nine and a half hours the oxide of copper will have been reduced to the metallic state, while the oxides of the baser metals, together with a certain amount of copper oxide, uniting with the siliceous ash of the coal, and with silica from the furnace bottom, will have passed into the state of slag.

This slag is carefully skimmed off during about an hour, and the metallic bath is exposed for another hour or an hour and a half to the strongly oxidizing action of the air. During this period, known as the *Verblasen*, sulphurous anhydride is evolved, together with zinc, lead, and some other metals, which, becoming oxidized, fill the furnace with dense fumes.

The next period, namely, that of roasting and sparkling, *Braten und Sprühen*, occupies from two to two and a half hours, during which further traces of various metals are eliminated, and a little sulphurous anhydride given off. The charge is now subjected to oxidation during two additional hours, at the end of which time it has assumed the condition of dry copper, equivalent to the *Gaarkupfer* of the old hearth-refinery.

The copper is then brought to tough-pitch by poling its surface covered by charcoal, and the metal laded into moulds. If during the process of lading the copper should exhibit any indication of becoming dry from absorption of oxygen, it is again poled for a few minutes.

The working of a charge, including refining, toughening, and lading, occupies from nineteen to twenty hours, every 100 tons of the material treated yielding from 68 to 69 tons of ingot copper of A. quality, and from 15 to 16 tons of slags and scoriæ, which are treated in the next operation.

IX. *Treatment of Smelting- and Refinery-Slags.*—The slags and the scoriæ resulting from the smelting and refining of first-quality copper are taken to the old liquation-works, where they are smelted in a small blast-furnace for blister-copper. The copper thus obtained is refined with additions of lead and phosphor-copper, toughened by poling, and sold as B. copper of second quality.

In 1884 the production of copper and silver at Mansfeld and Sangerhausen was:—

| | |
|---|---|
| Copper | 12,773 metric tons. |
| Silver | 74,132 kilos. |

PROCESSES FORMERLY EMPLOYED AT MANSFELD—*Rosette-Copper.*—The copper produced at Mansfeld was at one time largely delivered to commerce in the form of *Gaarkupfer* or rosette-copper, sold in the shape of thin discs, chiefly employed for making alloys, but not sufficiently soft for hammering and rolling.

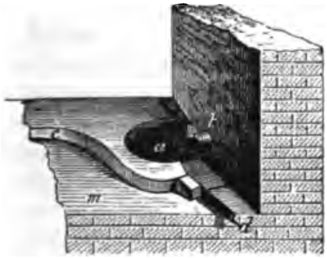

Fig. 129.—Kupfergaarherd.

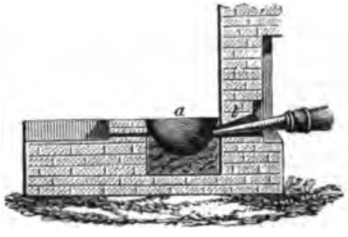

Fig. 130.—Kupfergaarherd; vertical section.

Blister-copper for the production of rosette-copper is treated in the small German hearth or *Kupfergaarherd.*

Fig. 129 represents a perspective view, and fig. 130 a vertical section, of this arrangement; it consists of a hemispherical basin, *a*, about 18 inches in diameter, excavated in a mass composed of four parts of pounded charcoal, four parts of fire-clay, and one of sand. This is surrounded by a low platform, *c*, level with the top of the basin, which is, on one side, furnished with a small door, *d.* When the hearth has been freshly lined, it is necessary, before proceeding with another operation, to dry it by the introduction of a few shovelsful of ignited charcoal, which is allowed to remain until the hearth is completely dry. As soon as this is the case, the cavity is filled with fresh charcoal, fragments of impure copper are arranged opposite the tuyer, *t*, and the blast is gradually admitted. When the first charge of crude metal has been thus melted, a further quantity is added, care being taken at the same time to supply a proper amount of fuel. The scoriæ formed during the progress of the operation escape through a tap-hole, which communicates with the cavity in which the refining is effected a little above the level of the top of the mass of masonry, *m*.

The first slags obtained are of a greenish colour, and contain a large quantity of oxide of iron. During the fusion, sulphurous anhydride, and sometimes arsenical vapours, are evolved.

The next slags are of a deep-red colour, and are rich in cuprous oxide. When the whole of the impure copper constituting a charge has been fused in successive small quantities, the workman takes samples, from time to time, by means of an iron rod, and, from the appearance of these, he is enabled to judge of the working of the furnace and the state of the metal it contains. As soon as the process is found to be sufficiently advanced, the blast is stopped and the surface of the metal is freed from fragments of charcoal. The slags are then carefully raked from the surface of the metallic bath, on which a little water is thrown to solidify the upper surface, which is at once withdrawn, by an iron hook, in the form of a thin circular plate. When the first disc has been thus removed more water is thrown on the surface of the metal, and a second film is coagulated and lifted off. These operations are repeated until the whole of the copper has been removed from the furnace.

The rosettes thus obtained do not exhibit the malleability and ductility of ordinary commercial sheet-copper, and in order to communicate to it these properties it is necessary to subject it to a final operation of toughening. For this purpose the rosettes are again melted in a similar furnace to that above represented; and as soon as the discs are fused, and have fallen into the small concave basin, the surface of the bath is sparingly covered with charcoal, by which, after a time, the oxide is reduced, and the metal attains a state of malleability.

*Liquation.*—The method of extracting silver from copper by liquation was employed in the Mansfeld district up to the year 1836. The efficiency of this process, which is described by Agricola, depends on the following principles. If lead and copper be fused together, the two metals unite; and if this mixture be rapidly cooled, after being run out of the furnace, they remain in a state of intimate admixture. If, on the contrary, the mixture be slowly heated to near its point of fusion, or be allowed to cool very gradually after being in a liquid state, the two metals will separate, and the lead will contain nearly the whole of the silver originally in combination with copper, while the latter metal retains only a comparatively small portion of the lead added. The silver may now be separated from the lead by cupellation, and the copper freed from that metal by refining.

Three parts of copper and from 10 to 12 parts of lead, already containing a certain proportion of silver—if such is to be procured—are fused together in a cupola-furnace; instead of metallic lead, litharge is sometimes employed. The fused alloy, on flowing from the furnace, is poured into cast-iron moulds, where it is rapidly cooled by the help of water, and from which it is removed in the form of large circular cakes.

These discs are subsequently heated on a liquation hearth, in order to extract the argentiferous lead in the liquid form, while the associated copper remains unmelted, and forms a porous mass, retaining the form of the original cakes.

This hearth, figs. 131 and 132, consists of two slightly inclined plates of cast-iron, so placed as to leave between them a space, S, beneath which is a hollow channel, C, left in the mass of masonry, M, which supports the iron plates. The metallic discs are placed perpendicularly on these, and are kept at a short distance from one another by means of wedges, while the open sides of the area are closed, after charging, by clay-lined plates of wrought-iron, F. The fuel employed, which is charcoal, is introduced between the metallic discs, after which the wedges are withdrawn, and some wood is placed in the channels, C, by the combustion of which the charcoal in the upper part of the hearth is ignited; the draught is produced by small chimneys, *d*, left in the masonry of the furnace. As the temperature of the cakes becomes more and more elevated, the lead, which is the more fusible metal, begins to melt, and, flowing over the surface of the iron plates, falls into the channels, C, and is conducted, by a depression in the floor, into the exterior basins, *b*. In proportion as these reservoirs become filled, the lead is removed, with an iron ladle, to a mould, where it receives the form of lenticular cakes. The copper, still retaining a certain amount of lead and silver, remains, in the form of half-fused spongy masses, in the position in which it was first placed. The lead thus separated by liquation contains a large portion of the silver, as, from the circumstance of the alloy of silver and lead being more fusible than the pure metal, a small quantity only of silver is retained by the lead which remains associated with spongy copper on the hearth of the furnace.

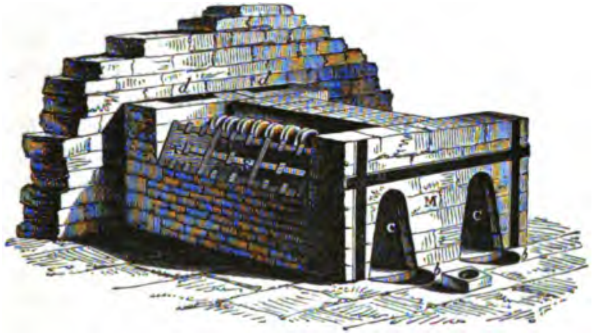

Fig. 131.—Liquation-Hearth.

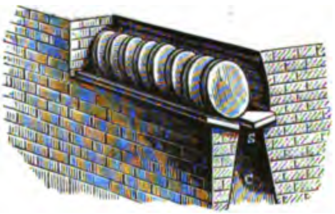

Fig. 132.—Liquation-Hearth; section.

These porous masses of copper are, however, still capable of affording a certain amount of argentiferous lead if submitted to a higher temperature, and for this purpose are heated in a peculiarly constructed apparatus, known as a *sweating-furnace.*

The spongy masses of copper, remaining after the liquation of the lead and silver, are charged on the hearth of the furnace, and rest on the brick piers by which the bottom is divided longitudinally into flues. These spaces are filled with wood, which is ignited, and the door closed. The draught is established through openings, in connection with a chimney, by which the smoke and heated air are carried off.

This treatment determines the separation, in a liquid form, of a further portion of lead, which becoming oxidized is chiefly converted into litharge, which falls to the bottom of the flues, together with a small quantity of oxide of copper dissolved in the oxide of lead. By operating in this way, copper is obtained still further freed from lead and silver than that coming from the liquation-hearth, and in the spaces between the piers will be accumulated litharge containing silver, and a small proportion of copper oxide. This mixture was employed at Mansfeld as a source of lead in the cupolas in which the fusion of the unrefined copper with lead was conducted.

The copper was formerly refined in a reverberatory furnace somewhat resembling the German cupelling-furnace hereafter to be described.

A method for the separation of silver from copper mattes by amalgamation was in operation at Mansfeld up to 1849.

## COPPER-SMELTING BY THE MANHÈS PROCESS.

This process, which has for some four and a half years been in operation at the copper-works of Eguilles, Department of Vaucluse, France, was described by the late Professor L. Gruner, whose paper for some time constituted almost the entire literature of the subject.[1]

Various persons have, at different times, speculated on the possibility of treating cupriferous substances by the process applied by Bessemer to the refining of cast-iron, without employing other fuel than the silicon, manganese, and carbon contained in the material itself. In the same way that iron ores are reduced in the blast-furnace to the state of pig-iron are copper ores reduced by a simple fusion to a coarse-metal composed of copper, iron, and sulphur. In both operations the metals are separated from their gangues, and in both are the most important products somewhat similar in constitution. In the one case it is a mixture of iron and manganese in combination with carbon and silicon, in the other a double sulphide of iron and copper. It would consequently appear probable that, if by simply blowing through the fused mass the carbon, silicon, and manganese can, in the one case, be eliminated, that, in the other, sulphur and iron might be removed by similar means, since both these elements are more readily oxidizable than copper.

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[1] 'Annales des Mines,' 8e Série, Tome iii. p. 429.

At first sight nothing would appear to be more easy, but nevertheless all the experiments, from the first, made in Russia in 1867, to those made in this country up to 1879 by Hollway, resulted in complete failure. In spite of their apparent similarity, the circumstances in the two cases differ materially.

In pig-iron the substances to be oxidized never exceed 10 per cent. of its weight, while copper matte often contains from 40 to 60 per cent. of copper; so that from 60 to 40 per cent. of its constituents have to be removed by oxidation. Moreover, the foreign bodies, silicon and carbon, present in cast-iron, develop a large amount of heat during their combustion; the first 7,800 and the second 8,000 calories, while the iron and sulphur of the matte yield but 2,200 and 1,500 calories respectively. It is true that the specific heat of copper is about two-thirds less than that of iron.

The weight of copper in copper mattes is, however, much less than is that of iron in cast-iron, so that in point of fact the reduction of copper matte requires less heat than the refining of cast-iron. *A priori*, then, the process does not appear impossible, although considerable difficulty might be anticipated from the large amount of iron to be oxidized and passed off in the state of scoriæ.

The experiments of M. Pierre Manhès were commenced in 1880, the apparatus employed being a small Bessemer converter of the usual construction, having an air vessel at bottom, with vertical tuyers injecting air from below upwards through the metal collected in the bottom. As might have been anticipated the sulphur and iron rapidly became oxidized, the former escaping as sulphurous anhydride, while the oxide of iron formed a liquid slag at the expense of the silica of the refractory lining.

At the commencement of each experiment the heat developed by the combustion of the iron and sulphur was sufficient to maintain the fluidity of the metallic bath, as well as that of the ferruginous slags. This quiet progress of the operation was not, however, long continued, and was followed by difficulties which manifested themselves towards the close of the experiment.

On the one hand, at a certain moment the progress of the operation suddenly changed, the regular ebullition produced by the injection of air was followed by rapid boiling, accompanied by violent projections of slag, this action ultimately becoming so intense as to entirely empty the vessel.

On the other hand, in proportion as the sulphur became burnt off, metallic copper being heavier than copper matte, accumulated in the bottom of the converter, where, from the disappearance of the combustible elements, it became chilled, and gradually obstructed the tuyers with solid copper.

In order to obviate the difficulty, resulting from this cooling action of the blast upon the reduced copper, the vertical tuyers were replaced by horizontal ones injecting the blast into the converter at a certain height above its bottom. In the apparatus thus modified the copper no longer became chilled, but, as soon as it was produced, fell into the bottom of the vessel, below the horizon of the tuyers, where it remained in a perfectly liquid condition until the close of the operation. In this way it was found possible to convert the whole of the matte into metallic copper not containing above 1 per cent. of impurity.

In order to overcome the inconvenience arising from the production of large quantities of ferruginous slags, numerous expedients had to be devised, but it was eventually found that when the mattes contained from 50 to 60 per cent. of copper the slags no longer presented great practical difficulties.

In 1881 this method of treating copper mattes was installed on a practical scale at Eguilles, Vaucluse, by the erection of three blast-furnaces, each 16 feet in height, for the fusion of ores, two cupolas 8 feet in height for re-melting the mattes, and three converters for the reduction of mattes into metallic copper. This plant was afterwards duplicated by the erection of a similar number of furnaces and converters. The capacity of each of these plants is equal to the production of from 90 to 100 tons of refined copper monthly.

The ores treated at Eguilles are very variable in composition, consisting of copper pyrites from Savoy, which is by no means rich, impure but tolerably rich ores from Aveyron and the Pyrenees, pure sulphides from Tuscany containing from 7 to 15 per cent. of copper, and ores from Algeria, some of which contain a considerable amount of antimony. These materials are frequently mixed with burnt cupriferous pyrites, copper-precipitate, furnace bottoms, &c.

Fig. 133 represents a vertical section of the converter employed at Eguilles, fig. 134 is a horizontal section through the tuyers, and fig. 135 a section at the level of the trunnions. Its greatest internal diameter is 4 feet 8 inches, its total height 8 feet, and the height of the tuyers above the bottom 1 foot. The inside of the condenser is lined with a mixture of fire-clay and siliceous sand, the blast being admitted through eighteen tuyers each $\frac{3}{8}$ inch in diameter, formed of perforated fire-bricks 9 inches in length.

The operation is conducted in nearly the same way as when cast-iron is operated on in the Bessemer converter. The vessel, previously heated to redness, receives a charge of 1 ton of fused matte, and the blast, at a pressure of from 10 to 12 inches of mercury, is turned on. The temperature rapidly increases, and thick white fumes of sulphurous anhydride escape into a hood in direct connection with a chimney 165 feet in height. After blowing for a period varying from fifteen to twenty minutes, according to the richness of the matte under treatment, white fumes cease to be evolved, and the contents of the converter are poured into moulds.

When the mattes operated upon are poor, the pouring off must be effected as rapidly as possible in order to avoid violent projections of the fused material. If, however, the mattes under treatment are moderately rich in copper, no difficulty of this nature is said to occur, and the product will be a material ready for the refinery, containing from 98 to 99 per cent. of pure copper, with slags of from 3 to 5 per cent., which go back to the blast-furnace. The lining of a converter usually requires renewal after from 16 to 18 blows.

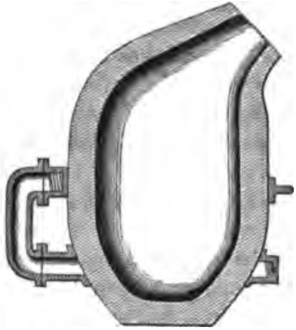

Fig. 133.—Vertical Section.

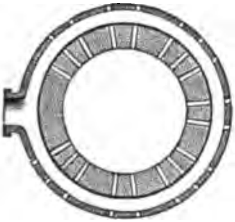

Fig. 134.—Section through Tuyers.

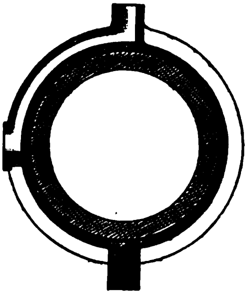

Fig. 135.—Section through Trunnions.

The Manhès Converter.

Mattes containing arsenic, antimony, lead, tin, and zinc, are said, when treated by this process, to yield copper of fair quality, since the foreign metals present are stated to be either oxidized or volatilized.

The copper obtained from the converter has a nearly constant composition, which varies but little from the following:—

| | |
|---|---|
| Copper . . . . . . . . | 98·5 to 98·8 |
| Sulphur . . . . . . . . | 0·9 „ 0·8 |
| Iron . . . . . . . . . | 0·6 „ 0·4 |
| | 100· |

Gruner estimated that at Eguilles a ton (1,000 kilos) of fine copper can be produced from 10 per cent. copper ores with an expenditure of

5 tons of coals, while he states that a ton of copper produced from the same ores by the Welsh method would require from 13 to 15 tons of a similar fuel. He further estimated that the total cost of producing a ton of copper at Eguilles is from £6, 8s. to £6, 16s., while he computed the cost of producing the same quantity of copper from similar ores by the Welsh method at from £12, 16s. to £15.

These figures, which are reproduced without comment, would, if correct, indicate an important advance in the treatment of copper ores. The establishment at Eguilles continues to employ this method of working, and licenses to operate under the various patents are stated to have been granted both in this country and in America.

In a paper recently published by Professor T. Egleston,[1] on the treatment of copper mattes at the works of the Parrot Silver and Copper Company, Montana, he states that when first invented it was supposed that the Manhès process would produce blister-copper from matte in one operation. This can be readily accomplished when matte containing 72 per cent. of copper is operated upon, but in practice it is generally found more economical to produce mattes containing about 40 per cent. of copper, and afterwards to blow them twice in the converter.

The process, as a whole, now consists of a variable number of operations depending on the extent of the plant available, and the number of blows to which the matte is to be subjected.

For poor ores the operations are the following:—

1. Melting the ores without previous calcination to obtain a matte yielding from 25 to 30 per cent. of copper.
2. Fusion of this matte in a cupola.
3. Blowing the fused matte in a converter until it contains 72 per cent. of copper.
4. Fusion of the resulting matte in a cupola.
5. Treatment of the fused matte in a converter for blister-copper.
6. Refining in a reverberatory furnace.

For rich ores the process, with a sufficient number of converters, requires operations 1, 5, and 6 only, as the mattes should be run directly from the cupola to the converter.

A more recent form of converter employed by M. Manhès consists of a cylindrical horizontal vessel of plate-iron lined internally with fire-brick, and provided, on one side only, with a row of tuyers. The depth of these below the surface of the molten charge is regulated by the partial rotation of the cylinder.

## WET PROCESSES FOR COPPER-EXTRACTION.

It has long been known that the waters issuing from certain copper mines contain a considerable amount of copper in the form of sulphate.

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[1] 'School of Mines Quarterly,' May 1885.

Agricola states that in his time the waters of a mine near Schmöllnitz, in Hungary, eroded iron and converted it into copper. Cupriferous waters of this description are plentifully discharged from mines of which the workings are extensive, and where sulphides of iron and copper are disseminated over extensive areas; these are converted by oxidation into sulphates of iron and copper, and the resulting solutions are sometimes sufficiently concentrated to enable the copper to be extracted with profit. This is done by bringing the waters in contact with either wrought- or cast-iron, which, abstracting the sulphuric acid from the copper, the latter precipitates as a crystalline powder, while sulphate of iron is carried off in solution; the *copper-precipitate* thus obtained is smelted and refined in the usual way. At San Domingos in Portugal and at Rio Tinto and Tharsis in Spain, enormous quantities of copper-precipitate are now annually produced.

At San Domingos the waters from the mines are conducted over large heaps of cupriferous pyrites, through which channels for the circulation of air have been made, and the liquors thus obtained are collected in reservoirs, where the copper is precipitated by pig-iron.

At Rio Tinto and Tharsis a portion of the pyrites is burnt in large heaps, *teleras*, previous to lixiviation, and the liquors obtained from the treatment of the burnt ore are used for leaching out the copper from a mixture of raw and spent burnt ores.

At Agordo in the Venetian Alps, at Mülbach in the Tyrol, and in certain localities in Norway, the process known as "Kernel-Roasting" has been employed for the treatment of poor copper ores. When cupriferous iron pyrites, in fragments about the size of the fist, is slowly roasted in heaps, it is found that a large proportion of the copper becomes concentrated, near the centre of the several lumps, in the form of a rounded nucleus consisting of a double sulphide of copper and iron.

This nucleus or "kernel," which gives its name to the process, is enclosed in a somewhat porous "shell," consisting, mainly, of ferric oxide, which can be easily detached by a hammer. The kernels are separated by hand, and smelted for copper, while the shells are lixiviated with water to remove the small amount of cupric sulphate which they contain. They are afterwards employed to form the beds upon which fresh heaps of cupriferous pyrites are calcined. At Agordo the raw pyrites contains on an average 1·60 per cent. of copper, the kernels 7 per cent., and the shells 0·70 per cent. of that metal. At Rio Tinto the ores when roasted in heaps frequently yield kernels containing 40 per cent. of copper.

The water from the Wicklow mines in Ireland also holds a small quantity of cupric sulphate in solution, and, in order to extract the copper, it is conducted through a series of troughs interrupted at intervals by deep tanks or hutches. In these troughs, which are inclined at an angle of from 7° to 10°, pieces of iron are so placed that the water flows constantly over them, and *cement-copper* is precipitated; this is from time to time swept down into the tanks, and at intervals collected and sold. The waters from the Parys Mountain mines, near Amlwch, likewise afford a certain amount of copper annually, and those of many mines in Cornwall and elsewhere are similarly treated for the copper they contain.

HYDROCHLORIC-ACID PROCESS.—In the vicinity of the village of Twiste, in Waldeck, several beds of sandstone, to a greater or less extent impregnated with green carbonate of copper, have long been known. This ore, although varying considerably in its produce, yields, on an average, from 1½ to 2 per cent., and was formerly raised and smelted, but this method of treatment not having produced satisfactory results, the operations were finally abandoned.

The insoluble nature of the quartzose gangue with which the copper is associated, suggested, some thirty years since, to Mr. Rhodius, at that time proprietor of the Linz metallurgical works, the possibility of treating such ores by hydrochloric acid, and an establishment for that purpose was erected about the year 1855.

The arrangement employed consisted of a crushing-mill for the reduction of the sandstone to a small size, sixteen dissolving-tubs to effect the solution, and a considerable number of tanks and reservoirs for the reception of the copper-liquors and the precipitation of the metal by iron. Each of the sixteen dissolving-tubs was 13 feet in diameter and 4 feet in depth, and was furnished with a wooden revolving agitator, set in motion by shafting connected with a water-wheel. This apparatus was sufficient for the treatment of 20 tons of ore daily, and the consequent production of from 6 to 8 cwts. of copper. In 1856, when the writer visited Twiste, the ore was raised and brought into the works at a cost of 4s. per ton, and each operation was completed in twenty-four hours—the liquors being removed from the tanks to the precipitating troughs by wooden pumps.

The acid employed at this establishment was procured from alkali-works in the vicinity of Frankfurt; it contained only 16 per cent. of real acid, and cost, delivered at the works, 2s. per 100 lbs. Each ton of sandstone operated on required 400 lbs. of acid, which was diluted with water down to 10 per cent. before being added to the ore. In order to precipitate 1 ton of copper, 1¼ ton of iron scrap was used, and the residues removed from the washing-vats after the operation retained but one-tenth of 1 per cent. of copper.

How long this process was successfully carried on we are not aware, but are informed that the works ultimately became unprofitable on account of a falling off in the yield of the ores.

This process for extracting copper from poor siliceous ores was for several years used at Alderly Edge, Cheshire, where it was introduced by Mr. Henderson.

The copper here occurs, chiefly as carbonate, in Bunter Sandstone, of which very large quantities were raised and treated; but the results obtained did not, at the ordinary price of copper, leave a large margin of profit.

**Longmaid's Processes.**—In the year 1842 Mr. William Longmaid took out a patent for "Improvements in Treating Ores and Minerals, and in obtaining Various Products therefrom, certain Parts of which Improvements are Applicable to the Manufacture of Alkali." This invention consists in roasting ground iron pyrites with common salt in a reverberatory furnace, by which sulphate of sodium is produced, while any copper that may be present is transformed into soluble cupric chloride. "The copper may be separated from the solution either with iron, as is well understood, or, as I prefer, by the addition of lime slaked in water, forming a milk of lime." The specification goes on to say: "The solution from which the copper has been separated may, if required, be concentrated by boiling, and set aside to crystallize in suitable vessels, very fine crystals of sulphate of soda being obtainable."

In the specification of a second patent, granted in 1844, for "An Improvement in the Manufacture of Copper, Tin, Zinc, and Peroxide of Iron," Mr. Longmaid makes the following observations: "I have discovered that there are circumstances under which, and situations where, ores containing copper, tin, and zinc, with sulphur, may with advantage be treated with common salt for obtaining the metallic parts, without depending mainly on the profits derivable from the sulphate of soda." The liquors obtained by the lixiviation of ores which have been furnaced with addition of common salt will contain various metals in solution, together with sulphate and chloride of sodium. "And I wish it to be understood that this invention is confined to treating ores containing copper, tin, or zinc. The copper contained in any liquor obtained as above explained may be precipitated, as is well understood, by means of iron, and the milk of lime may be subsequently employed for separating the zinc associated with an excess of lime and with some oxide of iron.

"The oxide of tin separates from the liquor by gravity with residuary matters, and if they be not broken fine enough for the washing process to separate the oxide of tin, they are to be broken before washing, to separate the tin in the ordinary manner. If the whole of the copper and zinc be not converted into the soluble form by the first operation, the insoluble residue may be treated with weak muriatic acid obtained by condensing that product (as is well understood) as it is evolved from the furnace where the ores are being treated with common salt, as above explained, or weak muriatic acid, otherwise obtained, may be employed to dissolve the copper and zinc not before rendered soluble in water, and these metals may be separated from the solutions thus obtained, as above explained."

This process was worked for several years at St. Helens, where the copper was first precipitated by iron, and the liquors subsequently evaporated down for salt-cake; it was abandoned about the year 1863.

At the works of Messrs. Allen, near Newcastle, it was likewise followed in its entirety for many years, but was ultimately so modified as to be applied only to the burnt pyrites obtained from vitriol kilns, and the recovery of sulphate of sodium was no longer attempted.

"HENDERSON'S PROCESS."—In 1860 Mr. William Henderson filed a specification for "Improvements in Treating certain Ores and Alloys, and in obtaining Products therefrom."

"These improvements relate, first to the treatment of copper and several other ores, when they exist as, or have been reduced or converted to the state of oxides, carbonates, or other salts of copper, or other metal, and especially when associated with silica and other matter insoluble in dilute acids.

"Secondly, to the treatment of ores of copper, lead, zinc, antimony, silver, cobalt, and several other metals, when they occur as sulphurets, mixed or singly, and combined with sulphur and iron, as iron pyrites, containing all or either of these metals, or partially calcined or burnt, and being then a mixture of oxides, sulphates, and sulphurets, with or without silica.

"And thirdly, to similar compounds when they exist wholly as oxides or salts, and associated with much silica."

Ores of the first class he proposes to attack, either with or without a preliminary roasting, by sulphuric acid, and to evaporate the resulting sulphate of copper to dryness in leaden pans, the anhydrous sulphate of copper being subsequently so heated as to drive off the sulphuric acid which is condensed in a leaden chamber. The resulting oxide is then mixed with carbonaceous matter and a small quantity of a siliceous ore, and is smelted in a reverberatory furnace in the usual way; the products will be blister-copper and slag free from regulus.

"The second class of ores I treat as follows: If the proportion of sulphur existing in the ore is more than one and a half times as much as the metal or metals to be extracted, it should be reduced to at least that amount by calcination, or if the ore contains much silica the proportion of sulphur may be even lower than an equal proportion. The ores are reduced to fine powder, the finer the better, and mixed with from 5 to 50 per cent. of common salt. The mixture is then placed in retorts or close calcining-furnaces, having flues or pipes communicating with the interior of the furnace or retort, and a condensing-apparatus. In these furnaces the mixture is subjected to various degrees of heat, according to the nature of the ore and the metal or metals contained in it. If the ore contains much sulphur and little silica, the heat must be applied carefully at first and gradually increased, stirring at short intervals. When the ore is one of copper or zinc, and has been previously burnt or calcined, the mixture of salt and pulverized ore may be at once subjected to a bright red heat; the volatilized chlorides passing into the condensing apparatus are condensed with water. The ore is withdrawn from the furnace whenever it ceases to smoke strongly, and if any copper or other metal still remains in the calcined ore it is only necessary to wash it with the hot acid solution that runs from the tower or condensing-apparatus; by these means the last trace of copper or other metal is readily extracted. The copper or other metal is obtained from these solutions by precipitation with iron, lime, or an alkali, and the sulphate of soda may be afterwards obtained by evaporation and crystallization. When a mixed ore is under treatment containing metals whose chlorides volatilize at different temperatures far removed, such as copper and tin, copper or zinc and arsenic, lead and antimony, I employ a furnace with two or more beds, all heated by the same fire or fires, each bed having a separate condenser. By these means the arsenic, tin,[1] or antimony are volatilized in the upper or colder beds and separately condensed, and the copper, lead, or zinc in the lower and hotter beds.

"The third class of ores I treat exactly as the second class, the silica taking the place of the sulphur and decomposing the salt at a red heat."

It will be seen that the method of treatment patented by Mr. Henderson for ores of the second and third classes does not materially differ from that of Mr. Longmaid, excepting that the former proposes to volatilize and subsequently to condense a large proportion of the copper.

The amount of copper thus volatilized is, practically, of no importance, but considerable quantities of chlorine and hydrochloric acid are evolved from the furnaces in which the calcination of the mixture of ore and salt is effected. These are condensed and advantageously employed for the lixiviation of roasted ores.

At the period when Mr. Longmaid was carrying out his invention the supply of ores suitable for his process was exceedingly limited, and was chiefly derived from the mines of Cornwall and Devon; by the time, however, his method, in a more or less modified form, had been incorporated into the patents of Mr. Henderson, Spanish and Portuguese cupriferous pyrites had found their way extensively into the English market. These, after being used as a source of sulphur, furnish an excellent material for such treatment, and numerous works have been established for extracting copper from "burnt ores" by calcination with salt and subsequent lixiviation.

TREATMENT OF BURNT CUPRIFEROUS PYRITES.—Very large quantities of cupriferous pyrites are annually sent into Great Britain from Spain and Portugal, but Norwegian pyrites is no longer extensively imported into this country. Spanish and Portuguese pyrites are approximately uniform in their composition, and the limit of variation in the amount of copper present may be taken at about 1½ per cent. A specimen of this mineral from the mines of San Domingos, in Portugal, which formerly supplied more than one-half of the cupreous pyrites consumed in this country, was found by Mr. F. Claudet to have the following composition:—

---

[1] When ores containing stannic oxide are thus treated chloride of tin is not formed.

| | |
|---|---|
| S | 49·00 |
| As | 0·47 |
| Fe | 43·55 |
| Cu | 3·20 |
| Zn | 0·35 |
| Pb | 0·93 |
| CaO | 0·10 |
| $H_2O$ | 0·70 |
| Siliceous residue | 0·63 |
| Oxygen and traces of various metals | 1·07 |
| | 100·00 |

This pyrites, after being burnt for the manufacture of sulphuric acid, leaves a residue representing about 70 per cent. of the weight of the raw ore, which is treated in the wet way for the copper it contains; an average sample of burnt San Domingos pyrites was found on analysis to have the following composition:—

| | |
|---|---|
| S | 3·76 |
| As | 0·25 |
| Fe | 58·25 |
| Cu | 4·14 |
| Zn | 0·37 |
| Co | traces |
| Ag | traces |
| Pb | 1·14 |
| CaO | 0·25 |
| $H_2O$ | 3·85 |
| O, loss, &c. | 26·93 |
| Insoluble residue | 1·06 |
| | 100·00 |

The treatment of these residues for copper comprehends the four following operations:—

I. Grinding and sifting.
II. Calcination with salt.
III. Lixiviation.
IV. Precipitation of the copper by iron.

I. *Grinding.*—For this purpose edge-mills or rolls are commonly employed, and, in order to insure uniformity of mixture, the salt is added to the burnt ore previously to grinding. Coarsely crushed rock-salt, added in the proportion of from 12 to 15 per cent., is generally used. Before, however, proceeding to grind the burnt ore, it is necessary to ascertain the amounts, respectively, of copper and sulphur which it contains, since on the relations existing between these bodies depends, to a great extent, the success of the operation. In the majority of cases the amount of sulphur should exceed that of the copper by about ½ per cent. The estimation of the former is effected by barium chloride, and the resulting sulphate of barium weighed; the latter is usually estimated by the use of a standardized solution of potassium cyanide.

When the proportion of sulphur in the burnt ore is less than that above stated, an addition must be made of finely ground unburnt pyrites; if, on the contrary, sulphur is present in excess, the ore must be mixed with other burnt pyrites from which the sulphur has been more completely expelled. The mixture of burnt ore, salt, and, when necessary, raw pyrites, is passed through a sieve having five meshes to the linear inch, and is then ready for the next operation.

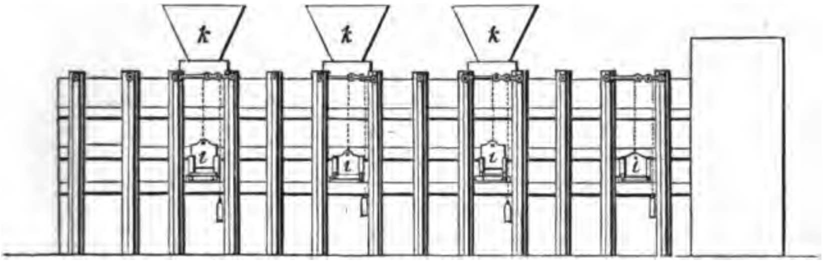

Fig. 136.—Roasting-Furnace; longitudinal elevation.

II. *Calcination.*—The furnaces in which this mixture is subjected to calcination vary considerably in their construction; at the various works belonging to the Tharsis Sulphur and Copper Company muffle furnaces, with fireplace at one end, are employed; at the Bede Metal Company's works at Jarrow-on-Tyne automatic furnaces, with revolving hearths, are to some extent used; but in many cases long muffle furnaces fired by gas are preferred.

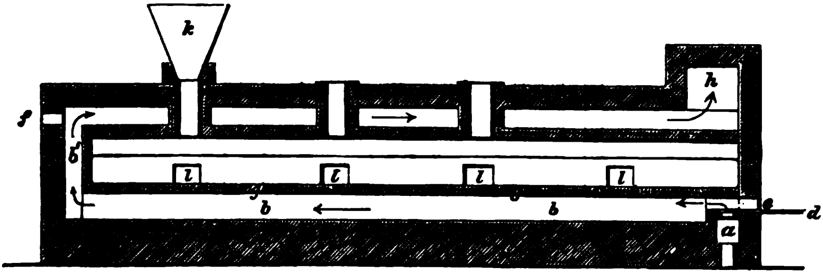

Fig. 137.—Roasting-Furnace; longitudinal section.

Figs. 136, 137, 138, and 139, represent the ordinary gas furnace in use in a great number of the Lancashire extraction-works. The first is a longitudinal elevation, the second a longitudinal section, and the third a horizontal section through the working-doors; fig. 139 is a transverse section through the centre of the fire-box, *a*. This furnace is 30 feet in length and 11 feet in width, outside measure. The gas from the producers is conveyed to the different furnaces through the flue, A, fig. 139, and entering the box, *a*, is admitted to the five parallel flues, *b*, separated from one another by brickwork pillars, *c*, the supply being regulated by dampers, *d*, fig. 137. A sufficient amount of air to consume a portion of this gas is admitted by apertures, *e* (fig. 137), at the end of each flue, which can be closed by sliding doors, while the portion which remains unconsumed is burnt by the aid of a further supply of air admitted by the openings, *f*, into the space, *b′*, situated at the opposite extremity of the five flues. In this way a long flame is made to travel under the bottom of well-jointed and rebated tiles, *g*, and over the brick arch of the muffle. By this means a tolerably equable temperature is maintained throughout the whole length of the apparatus, and the products of combustion escaping by *h*, are conveyed to the main flue, I, and pass direct to the chimney.

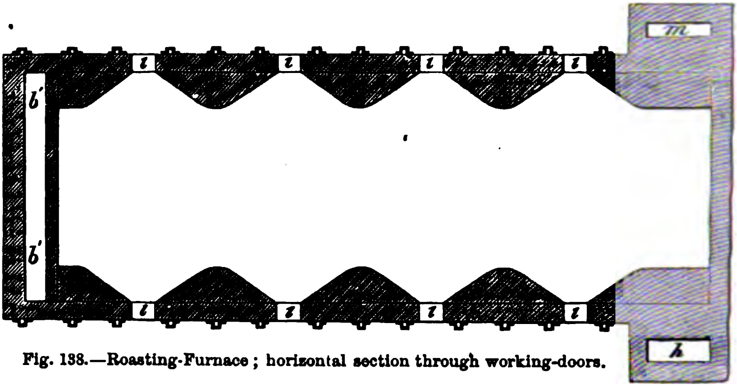

**Fig. 138.—Roasting-Furnace; horizontal section through working-doors.**

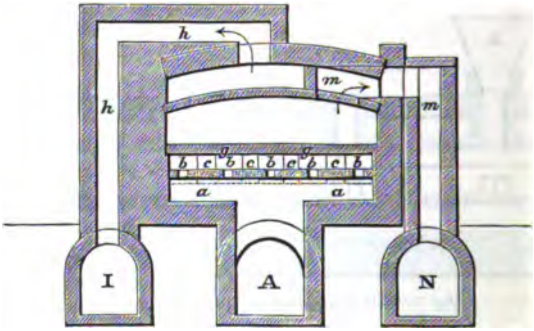

**Fig. 139.—Roasting-Furnace; transverse section through fire-box.**

The gases and vapours evolved from the charge are carried by a flue, *m*, to the flue, N, and pass through an ordinary condensing-tower, filled with coke, on their way to the chimney.

The mixture of ground ore and salt is charged, by means of a high-level railway, through the hoppers, *k*, and is at once evenly spread over the surface of the hearth, where it is kept at a dull red heat and frequently stirred by paddles or rakes, through the doors, *l*, until an assay indicates that it is in a fit state for drawing. This is determined in the following way:—A fair sample of the charge is obtained by removing a small portion with a paddle from every part of the hearth; about one ounce of this is taken, and after being finely ground in a cast-iron mortar, it is boiled with water, which is subsequently poured off. This operation is repeated, successively, three times, and the residue afterwards boiled with dilute hydrochloric acid, which is in its turn poured off, and excess of ammonia added to it; finally, the residue from hydrochloric acid is boiled with nitric acid, and ammonia added in excess. If it be found by these trials that the attack by hydrochloric acid only acquires a slightly blue tint, and that the nitric acid solution is entirely, or very nearly, colourless, the charge is ready to be drawn, since it indicates that, practically, the whole of the copper has been rendered soluble in water. When, on the contrary, the nitric acid solution contains copper, it shows that calcination has not been continued for a sufficient length of time, and should the hydrochloric acid solution, on the addition of an excess of ammonia, become decidedly blue, it is probable that the amount of sulphur present in the charge is not sufficient to effect the necessary chemical transformations.

During the process of roasting, the sulphur becomes oxidized, and sulphate of sodium and soluble cupric chloride are formed; unless, however, the charges are properly compounded and carefully worked at a suitable temperature, a large quantity of comparatively insoluble cuprous chloride will be produced, and a serious loss of copper in the residue will be the result. The charge weighs 3 tons 5 cwts., and the time required for furnacing is about six hours; the hydrochloric acid and chlorine which are evolved, together with a small quantity of iron and copper chlorides, are passed into a condensing-tower, through which a constant shower of water is caused to descend. These waters, which are somewhat acid, are employed for the lixiviation of calcined ore, and, in addition to hydrochloric acid and chlorine, contain iron and traces of copper.

III. *Lixiviation.*—The calcined mixture of burnt ore and salt is raked from the furnace, through the doors, *l*, in one of its sides, and while still hot, is charged into lixiviating tanks, which are usually from 10 to 11 feet square, and somewhat less than 4 feet in depth. These are made of wood tightly calked, and are provided with a removable false bottom, either of perforated tiles or of fire-bricks, supported on other bricks resting, on edge, on the floor of the tank. Upon this is placed a layer of cinders 3 inches thick, forming a coarse filter, on which are charged about 15 tons of the calcined ore. The plug-hole between the bottom of the tank and the filter bottom is now closed, and either hot water, *weak liquors*, or *tower-liquors* are run in, until the surface of the ore has been covered to a depth of several inches. This is allowed to remain about two hours, when it is run off into proper receivers, and the tank again filled with hot water, weak liquors, or tower-liquors. In this way each tank receives from nine to ten successive washings, and at about the seventh washing a little hydrochloric acid is sometimes added. All the liquors which drain from the several tanks after the addition of acid are collected in a separate receiver, and, instead of being run into cisterns containing iron scrap for the precipitation of the copper, they are employed under the name of "weak liquors" for washing the next series of tanks filled with freshly furnaced ore. The operation of washing occupies about forty-eight hours, and the residual *purple ore*, which is sold for "fettling" puddling-furnaces, as well as for other purposes, and of which an analysis is given, p. 292, should not contain above 0·15 per cent. of copper. The testing of the purple ore is conducted in a similar way to that of the furnaces.

Considerable quantities of fume are deposited in the flues connecting the various furnaces with the condensers; these are from time to time cleaned out, and the deposit treated for the copper it contains. A sample of dry dust, obtained in cleaning a flue at the Widnes Metal Works, in connection with eight calciners working on San Domingos ores, was found to have the following composition:—

| | |
|---|---|
| $As_2O_3$ | 2·97 |
| $Bi_2O_3$ | 2·21 |
| $Sb_2O_3$ | 0·23 |
| $ZnO$ | 4·06 |
| $Fe_2O_3$ | 15·98 |
| $PbO$ | 2·00 |
| $CuO$ | 25·33 |
| $Al_2O_3$ | trace |
| $CaO$ | 2·38 |
| $MgO$ | trace |
| $K_2O$ | 0·23 |
| $Na_2O$ | 1·81 |
| $NaCl$ | 2·73 |
| $SO_3$ | 40·32 |
| $SiO_2$ | trace |
| | 100·25 |

This analysis is chiefly interesting as showing the large number of metals contained in the ores treated.

IV. *Precipitation.*—In establishments in which the silver is extracted from burnt pyrites by Claudet's process, the liquors from the first three washings, which contain, practically, the whole of that metal, are first treated with a soluble iodide, and the iodide of silver formed is allowed to settle. The copper-liquors drawn off from the iodide of silver are, in such cases, added to the weaker solutions from subsequent washings; but when the silver is not thus separated the whole of the copper solutions, with the exception of the weak liquors, are at once run directly into the precipitating-tanks. These are commonly 12 feet square and 4 feet in depth, and are partially filled with clean iron scrap; precipitation of the copper being accelerated by boiling the liquors by the introduction of a jet of steam. In the course of some twelve hours complete precipitation of the copper will have taken place, and, when stains of that metal are no longer deposited on a brightly polished knife-blade when dipped into the liquors, they are first allowed to settle, and subsequently run off.

In order to separate the precipitated copper from the undissolved iron the mass is turned over with an iron fork, and the larger pieces of that metal picked out. The remainder is now washed on perforated cast-iron plates, through which the particles of precipitated copper pass into a properly constructed receiver; the fragments of iron are raked off their surface and returned to the precipitating tanks. The *precipitate* thus prepared will contain from 60 to 80 per cent. of metallic copper, and may be either melted directly for blister-copper, or, which is preferable, fused with sulphides, and afterwards subjected to a process of roasting.

MODIFICATIONS OF THE ORDINARY WET PROCESS.—In some establishments, instead of precipitating the copper by the use of iron scrap, *sponge-iron* has been employed. This is prepared by heating a mixture of coke-dust and purple ore in a reverberatory furnace, and drawing the reduced metal into cast-iron vessels, which are hermetically closed until they and their contents have become nearly cold. When sufficiently cooled, the spongy metal is ground under edge-runners, and sifted through a fine sieve. The precipitation of copper is very rapidly effected by the use of iron sponge, but the precipitate obtained invariably contains a considerable percentage of iron.

The amount of sulphate of sodium produced during the treatment of burnt pyrites in the wet way is very considerable, and various attempts have, at different times, been made to effect its recovery by crystallization or otherwise. At the Bede Works the waste liquors were for some time converted into soda-ash by a process invented by Mr. T. Gibb, the then manager; sulphate of sodium in the waste liquors was converted into sodium sulphide by balling with coal-dust, and carbonic anhydride passed through the liquors obtained by lixiviation. Carbonate of sodium was thus formed, and the sulphuretted hydrogen evolved was employed for the precipitation of copper from the solutions obtained by washing calcined ores. The processes adopted were, however, of a complicated nature, and this method of treating the liquors has been long since abandoned.

## COPPER-REFINING BY ELECTROLYSIS.

In order to obtain chemically pure copper, and, at the same time, to recover the silver and gold which is present in notable quantities in many varieties of bar-copper, the process of refining by electricity is sometimes employed.

In the year 1866, Messrs. Elkington of Birmingham obtained letters patent for a process of this description in which impure copper, in connection with the positive pole, is placed in a suitable solution, while in proportion as this impure copper dissolves, pure copper is deposited on the negative pole. The specification further states that a series of tanks containing a nearly saturated solution of sulphate of copper is to be employed, and that a magneto-electric machine is to be preferred as the source of electricity.

Copper-refining by electrolysis has now for some years been extensively carried on by the Norddeutsche Affinerie at Hamburg, by MM. Œschger and Mesdach at Biache-Saint-Waast, by M. Hilarion Roux at Marseilles, at the Oker Works, and by the Mansfeld Mining Co. in Saxony, by Lyon-Almand & Co., Paris, by M. André in Frankfurt, and by sundry refiners in this country, among the principal of whom are Messrs. Elkington & Co., and Messrs. Elliott's Metal Co., Limited, of Birmingham.

As the establishments in which refining by electricity is carried on are not easily accessible to strangers, it is impossible to obtain precise information relative to all the details of the various operations. The Norddeutsche Affinerie at Hamburg is conducted upon a very extensive scale, and the copper produced in that establishment has the reputation of being of exceptionally good quality. The following data relating to that and some other refineries are given on the authority of a recent work by M. Hippolyte Fontaine.[1]

The currents employed at the Norddeutsche Affinerie are obtained from six large Gramme machines and from another very large dynamo specially constructed for these works.

Pure copper to the amount of 2½ tons is produced daily, and a very large amount of gold is stated to have been annually recovered from the muddy residues deposited in the baths.

The large machine before mentioned makes 500 revolutions per minute. The number of baths is 40, arranged in two series of 20 each; the surface of the anodes in each bath is 323 square feet, thus making a total of 12,920 square feet. The cathodes, of fine copper, are about $\frac{1}{25}$ inch in thickness, the distance between the anodes and cathodes is 2 inches, and the daily production of refined copper 1,760 lbs. avoirdupois. This machine has worked continuously night and day during the last nine years, with an expenditure of 16-horse power. Two other series of tanks are more economically worked with regard to the power employed. In these two series there are 120 baths, arranged for tension, each having an anode surface of 161 square feet. The current is supplied by two Gramme machines, coupled for tension, requiring 12-horse power, and depositing 1,980 lbs. of copper every twenty-four hours.

At the works of MM. Œschger and Mesdach, a machine is employed in all respects similar to that made for the Norddeutsche

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[1] 'Électrolyse par Hippolyte Fontaine, Librairie Polytechnique,' Paris, 1885.

Affinerie at Hamburg, and furnishes a current to 20 baths, yielding a production of 880 lbs. of copper daily. Each bath contains 88 anodes and 69 cathodes, of which the total surfaces are equal. The anodes have a thickness of $\frac{3}{8}$ inch, and the cathodes, which are of pure copper, are $\frac{1}{25}$ inch in thickness. If both surfaces of the anodes be measured their united area will be 12,900 square feet. The distance between the anodes and cathodes is 2 inches, and the copper is deposited upon the latter in the form of thick plates, sufficiently sound to be taken directly to the rolling-mill. Frequently, however, the refined copper is detached from the cathode, which is readily accomplished.

The gold and silver fall to the bottom of the tanks in the form of a muddy deposit, together with such pieces of copper as may, from time to time, become detached from the plates forming the poles. The baths, when required, are emptied by a leaden syphon, and the mud is washed and passed through a sieve to separate any pieces of copper which may be present. After drying, this deposit is fused with litharge, and the resulting lead subjected to cupellation.

At the Oker Copper Works, near Goslar, three large dynamos have been put up by Messrs. Siemens & Co., one of which has been working continuously during the last four years, but the two others are of more recent construction. Each of these machines supplies a current to 10 or 12 large tanks, and precipitates daily from 528 to 660 lbs. of copper, with an expenditure of from 10- to 12-horse power.

M. Fontaine gives the following estimate of the cost, per ton, of refining copper by electricity at various works, but is careful to call attention to the fact that it must be regarded only as a very rough approximation.

COST OF REFINING COPPER BY ELECTRICITY.

| Name of Works. | Interest on Capital. | | | Motive Power. | | | Repairs. | | | Labour. | | | General Expenses. | | | Totals. | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | £ | s. | d. | £ | s. | d. | £ | s. | d. | £ | s. | d. | £ | s. | d. | £ | s. | d. |
| Hilarion Roux, Marseilles | 3 | 3 | 0 | 4 | 9 | 8 | 0 | 15 | 0 | 2 | 17 | 8 | 4 | 6 | 8 | 15 | 12 | 0 |
| Norddeutsche Affinerie, Hamburg | 2 | 11 | 9 | 1 | 11 | 7 | 0 | 9 | 7 | 1 | 12 | 0 | 1 | 12 | 1 | 7 | 17 | 0 |
| Elliott's Metal Co., Birmingham | 1 | 8 | 9 | 7 | 4 | 0 | 1 | 4 | 0 | 2 | 6 | 3 | 2 | 6 | 3 | 14 | 9 | 3 |

As an example of the extent of plant required for this method of refining it may be stated that in the Hamburg establishment there are no less than 12,800 square feet of anode, and an equal surface of cathode, in operation at the same time; and that the total weight of copper constantly in the baths is 125 tons.

Copper-refining by electricity is consequently an expensive operation, necessitating an extensive plant and a considerable amount of unproductive stock. On the other hand, the copper obtained is purer, and fetches a higher price, than that produced in the ordinary way, while the whole of the gold and silver contained in the copper operated on is recovered and made available.

## TREATMENT OF COPPER MATTES BY ELECTROLYSIS.

By the Marchese process the mattes operated upon are cast into plates so as to form anodes, while the cathodes are thin sheets of copper; these are immersed in baths containing from 3 to 4 per cent. of copper obtained by roasting and lixiviating rich copper mattes. At Stolberg, where mattes containing copper, lead, and silver have recently been subjected to electrolysis upon a working scale, their composition is approximately as follows:—

| | |
|---|---|
| Cu . . . . . . | 15 to 16 per cent. |
| Pb . . . . . . | 14 „ |
| Fe . . . . . . | 41 to 42 „ |
| S. . . . . . . | 25 „ |
| Ag . . . . . | 16 oz. 7 dwts. per ton. |

These mattes are obtained by roasting and re-melting the coarse-metal resulting from the treatment of argentiferous lead ores containing copper.

The dynamo employed is by Siemens & Halske of Berlin, belonging to their class $C_6$, making 1,118 revolutions per minute, and consuming somewhat less than one horse-power. The copper, which is deposited at the rate of 29·30 lbs. avoirdupois in twenty-four hours, is chemically pure, while the lead and silver, which either remain on the anodes or accumulate in the form of mud at the bottom of the tanks, are treated by one of the ordinary methods employed for the separation of those metals.

The above experiments, upon a moderate scale, having been regarded as satisfactory, the "Actien Gesellschaft von Stolberg und Westphalen" has proceeded to erect works capable of producing from 12 to 13 cwts. of copper in twenty-four hours from their own and purchased mattes. Ultimately M. Marchese hopes to be enabled, from such mattes, to deposit a ton of copper in twenty-four hours with an expenditure not exceeding 45 horse-power.

This process has been carried on during the last eight months (1885) at Sestri Levante, near Genoa, by the "Società di Miniere di Rame e di Elettrometallurgia," on mattes containing 20 per cent. of copper, and another establishment was subsequently started at Ponte San Martino, Piedmont, on very similar material. The process employed is similar to that in operation upon the plumbiferous mattes of Stolberg.

## ALLOYS OF COPPER.

The addition of zinc materially affects the colour of copper; if added in small proportion, the alloy assumes a golden-yellow colour; if the percentage of zinc be greater a pale-straw colour is obtained, and if zinc predominates, the colour of the alloy is greyish-white or iron-grey.

Various alloys of this kind, of which the most important is known by the name of *brass*, are employed in the arts. The proportions of the two metals best calculated for the production of fine brass seem to be about two parts by weight of copper to one of zinc.

*Brass solder* consists of two parts of brass and one of zinc, to which a little tin is occasionally added; but when the solder is required to be very strong, as for uniting the edges of tubes intended subsequently to undergo the process of drawing, two parts of common brass and two-thirds of a part of zinc may be employed. *Mosaic gold* consists, approximately, of 65 parts of copper and 35 of zinc. *Bath-metal* is composed of about 78 parts of copper and 22 of zinc. *Pinchbeck* and *Mannheim gold* are merely different names for an alloy similar to *Prince's metal*, which is composed of three parts of copper and one of zinc, separately melted in different crucibles, and afterwards mixed and incorporated by stirring.

Copper is sometimes externally converted into brass by exposure, when at a red heat, to the vapour of zinc; in this way are prepared the copper bars from which the so-called "gold wire" of Lyons is manufactured.

Copper is also extensively alloyed with tin, in combination with which it yields many valuable compounds, variously named in accordance with their respective compositions and uses.

*Gun-metal*, of which cannon are made, consists of, about, copper 91, tin 9.

*Bell-metal* is composed of, about, copper 78, tin 22.

The alloy of which gongs and cymbals are manufactured has usually the following composition: copper 80, tin 20.

*Bronze.*—Alloys of copper and tin, although still important, were more so before iron was extensively used, since prior to that period they were employed in the manufacture of cutting instruments. For this purpose a mixture of 90 parts of copper to 10 of tin was most commonly used, although a little lead was occasionally added, perhaps with a view of imparting to the alloy a certain degree of toughness.

The preparation and fusion of these different mixtures is, according to the quantities required, conducted either in a reverberatory furnace, or in strongly heated crucibles.

*German silver*, an alloy of copper, zinc, and nickel, has been long known and extensively used in China, and was formerly imported into Europe under the name of *packfong*. In 1776 it was first recognized as an alloy of copper and zinc with nickel, and from being afterwards largely manufactured in Germany it received the name of German silver. It is now extensively prepared in various parts of England, but particularly at Sheffield, where it is manufactured into spoons, forks, and various other articles for domestic use; these are plated by the electrotype process, and employed as a substitute for silver.

The following analyses, given by Dr. Lamborn, serve to show the composition of several varieties of this alloy:—

| | Cu. | Ni. | Zn. |
|---|---|---|---|
| Chinese packfong | 43·8 | 15·6 | 40·6 |
| English German silver | 61·3 | 19·1 | 19·1 |
| Berlin argentan | 52·0 | 26·0 | 22·0 |
| Sheffield German silver | 57·0 | 24·0 | 13·0 |

*Phosphor-bronze* is an alloy particularly well-suited for making fine and sound castings; a good specimen of this material was found on analysis to have the following composition:—

| | |
|---|---|
| Cu | 82·20 |
| Sn | 12·95 |
| Pb | 4·28 |
| P | 0·52 |
| | 99·95 |

The electric conductivity of phosphor-bronze is only about 20 per cent. that of copper, but silicon-bronze conducts electricity almost as well as copper itself.

*Manganese-bronze* is prepared by combining ferro-manganese, in different proportions, with various bronze and brass alloys, thus producing qualities suitable for special purposes. This alloy is employed as a material for the propelling-screws of steamers.

*Delta-metal* is an alloy of copper, zinc, and iron, prepared by adding to melted copper the alloy of zinc and iron which is formed in vessels in which the latter metal is externally covered with the former by the process known as "galvanizing." When fused it is said to run freely, and to afford fine-grained and perfectly sound castings, which in colour somewhat resemble pale gold. It is easily rolled into sheets or bars, and can be drawn into wire. Under favourable circumstances this alloy is said to withstand a breaking strain of 33 tons per square inch.

Although the foregoing are some of the more important alloys of copper, there are numerous others which are occasionally employed by the artisan. With iron it appears to combine in very small proportions only, with aluminium it forms an alloy of considerable malleability and great hardness, which is capable of taking a high polish.

**Brass.**—Brass, for which the old name is *latten*, is essentially an alloy of copper and zinc.

The first brass-works erected in England are said to have been put into operation in 1649 at Esher, in Surrey, where rosette-copper, imported from Sweden, was exclusively employed in the manufacture; the proprietor having, however, become involved in a disastrous lawsuit, the establishment was ultimately broken up. Birmingham, where the trade is stated to have been first introduced in 1740, by the Turner family, is now the principal seat of the brass industry of this country. Brass is harder than copper, and consequently better calculated to resist wear; it is also in a high degree malleable and ductile, so that it is easily rolled into sheets and readily hammered into vessels of any required shape. It can, moreover, be worked by the process of stamping into numerous ornamental and useful objects, and admits of being drawn into fine wire; it fuses at a lower temperature than copper, and is capable of receiving a more delicate impression of the mould. Finally, it turns easily in the lathe, its colour is agreeable, it is capable of receiving a high polish, and it possesses over copper the advantage of greater cheapness.

Until a comparatively recent date brass was exclusively made by the old process of cementation, which has become superseded by directly alloying copper with metallic zinc.

The general name of brass is applied to alloys of copper and zinc, into the composition of which the two metals enter in very different proportions. The following table, compiled from Mr. Mallet's figures (Report of the Meeting of the British Association at Glasgow, 1839), gives the proportions and peculiarities of several varieties of brass:—

| Cu. | Zn. | Colour of Alloy. | Fracture. | Remarks. |
|---|---|---|---|---|
| 88·60 | 11·40 | Reddish-yellow | Finely crystalline. | |
| 83·02 | 16·98 | Yellowish-red | „ | |
| 79·65 | 20·35 | „ | „ | Rolled brass. |
| 74·58 | 25·42 | Pale yellow | „ | Mosaic gold. |
| 66·18 | 33·82 | Full yellow | „ | German brass. |
| 49·47 | 50·53 | „ | Coarsely crystalline. | German watchmaker's brass. |
| 31·52 | 68·48 | Silver-white | Conchoidal | Very hard and brittle. |
| 24·50 | 75·50 | Ash-grey | Finely crystalline | Brittle. |

*Manufacture of Calamine Brass.*—This very ancient process, by which for a long period brass was exclusively manufactured, seems to have disappeared from among the industries of the United Kingdom. The various operations of this process are conducted in the following way:—The furnace employed consists of a circular chamber lined with fire-bricks, contracted above to a circular opening serving as a chimney; the bottom is closed by a cast-iron plate, in which are twelve holes, symmetrically arranged around one larger hole in the centre. Through this central hole are withdrawn the ashes and clinkers, which fall into an ash-pit, communicating, by means of an arched air-way, with a long passage or vault by which air is conveyed to the furnace from the outside, and through which access to the ash-pit is obtained. Over the holes in the bed-plate, with the exception of that in the centre, are placed cast-iron tuyers or nozzles, 6 inches in length, 2 inches in diameter at bottom, and 1 inch at top, inside measure; the space between the nozzles is filled up level with their upper extremities with refractory bricks set in fire-clay, so as to form a level floor 6 inches in thickness. This forms a substitute for the ordinary fire-grate, the air necessary for sustaining combustion entering through the different nozzles. Several of these furnaces are usually constructed in a row, and over the whole is built a brickwork chamber terminating in a cone open at the top like that of an ordinary glass-house. These furnaces have no chimney excepting the mouth, which is kept more or less closed by a sliding cover consisting of a circular fire-tile set in an iron framing.

The crucibles employed are round, and made of fire-clay, each being 12½ inches in height and 8½ inches wide at top; the central crucible, sometimes called the *king-pot*, is often a little larger than the others, being capable of holding 120 lbs. of metal, whereas the others contain only 84 lbs. each.

A mixture of 100 lbs. of finely-ground and well-calcined calamine or roasted blende and 40 lbs. of ground coal, is intimately incorporated, dry, and then passed through a sieve of eight holes to the linear inch. This mixture is subsequently damped and then passed through a much coarser sieve, after which it is mixed with 66 lbs. of granulated copper, *bean-shot*, and is then ready for charging.

As the operation of making brass is carried on continuously, the mixture is introduced into the crucibles while they are still red hot from the treatment of a previous charge, and their mouths are severally covered by large pieces of coal, while the spaces between the different crucibles are filled with coal broken into pieces of the size of the fist. The mouth of the furnace is now partially closed for one hour and a half, for the purpose of transforming the coal into a kind of coke; the orifice is then still further closed for a short time, and the coke properly arranged between the different pots, care being taken to keep all the air-holes open. The heat is now progressively raised by the gradual removal of the cover, and, if skilfully conducted, the operation will be completed in about ten hours.

In order to collect the brass which has been formed, the king-pot is first taken out, and its contents well stirred with an iron rod flattened at the end; one of the side pots is next removed, treated in the same manner, and the brass, which collects at the bottom, poured into the king-pot. All the other side pots are successively treated in the same way, and the brass which has been collected in the central pot is finally skimmed and poured into moulds. Dr. Percy states, on the authority of an old calamine-brass maker, that good pots lasted, on an average, sixteen days, and were not allowed to cool during that period.

*Direct Preparation of Brass.*—This may be effected by melting together a mixture of copper and zinc, either in crucibles or in a reverberatory furnace. When this operation is conducted in crucibles the zinc should be added to the copper immediately after the latter has entered into fusion, and the ingots of copper should be heated to redness previously to their introduction into the pots. In making castings, and in the re-melting of brass, there is always a considerable loss of zinc through volatilization, for which allowance must be made when arranging the mixture. Granite moulds were formerly used for casting ingot-brass, but iron is now generally employed.

Muntz's metal, or yellow-metal, which has almost entirely superseded copper-sheathing in the merchant service, is prepared in reverberatory furnaces, the zinc being gradually added to the melted copper. Before tapping, samples of the alloy are taken out of the furnace in the same way as the copper proofs made use of in the process of refining; these are cast into oblong ingots, hammered, and broken in a vice, the quality of the mixture being judged of in accordance with the appearance presented by the fracture. This should be close and finely granular, but if the first trial should not prove satisfactory more zinc or copper is added, and the mixture well stirred; this is repeated until the fracture of a sample ingot indicates that the right proportions of the two metals have been reached. The necessity for this method of testing is caused by the great facility with which zinc becomes volatilized, and consequently, although the proper quantities of the two metals may have been charged into the furnace, it is impossible to make accurate allowance for the amount of zinc which may be driven off. It also frequently happens that the charge of a furnace is made up with a mixture of old yellow-metal, new copper, and zinc; in such cases the difficulty of making due allowance for loss of zinc will be still greater. When ready the metal is tapped into a large ladle and either poured or laded into closed cast-iron moulds, the interiors of which have been either oiled and dusted with charcoal, or, more commonly, washed with a mixture of wood-ashes, or clay, with water.

Yellow-metal may contain from 50 to 63 per cent. of copper, and from 50 to 37 per cent. of zinc.

From the circumstance that tin is not a very volatile metal, the preparation of bronze is a more easy operation, and may be conducted either in crucibles or in a reverberatory furnace.

## TIN.

Tin is a white metal, with a lustre closely approaching to that of silver, and with a specific gravity of 7·29; it possesses a characteristic odour which becomes evident when a piece of this metal has been slightly warmed by being held for some time in the hand. It is very malleable, and may consequently be reduced to thin leaves by hammering; it occupies the fourth rank in order of malleability, and the eighth in point of ductility. Tin, although flexible, is not elastic, and when bent emits a peculiar crackling sound, which is most distinct in the purest specimens; a perceptible elevation of temperature is caused by the repeated bending and straightening of a bar of tin. It melts at a temperature of 227·8° C., and when very strongly heated, gives off distinct fumes, and experiences a loss of weight. Tin exhibits a great tendency to crystallize; this property may be readily made apparent by slightly attacking its surface by an acid capable of removing the exterior. When this has been done, the metal assumes a mottled appearance, caused by the irregular reflection of the fern-like crystals brought to light by the action of the acid. A process of this kind is sometimes resorted to for improving the appearance of articles made of tin-plate, which, after being treated by weak hydrochloric acid, and subsequently covered by a coating of transparent coloured varnish, present a variegated and prettily marked surface.

Tin may be obtained in the form of crystals, by fusing a considerable weight in a ladle, or crucible, and allowing it to cool gradually on a heated sand-bath; as soon as a solid pellicle has formed on the surface, it is pierced by a hot iron bar, and the internal portions, which still remain in a liquid state, are allowed to run out. By operating in this way crystals of considerable size, though rarely exhibiting very sharp and well-defined edges, will be found lining the cavity from which the liquid metal has been removed. Tin may be deposited from its solutions in a crystallized state by electrical agency, and can by this means be obtained in the form of brilliant elongated needles. This metal is probably dimorphous, and appears to crystallize both in cubes and in tetragonal prisms.

The tin of commerce is never quite pure, but is more or less contaminated by the presence of various other metals, particularly arsenic. To obtain tin in a state of extreme purity, granulated tin may be attacked by strong nitric acid, and the resulting insoluble residue washed, first with hydrochloric acid and subsequently with hot water. The white residual powder is now reduced to the metallic state by fusion, with the addition of a little charcoal, in a lined crucible. Tin is but slightly affected by exposure to the air at ordinary temperatures, but when fused its surface is rapidly covered by a crust of greyish colour, consisting of a mixture of metal and stannic oxide. This oxidation of tin takes place very rapidly at high temperatures, and when the metal is heated to whiteness, is attended by distinct combustion; at a full-red heat it decomposes water, with evolution of hydrogen gas. It is dissolved in strong hydrochloric acid, and its solution is also effected by warm dilute sulphuric acid; concentrated sulphuric acid, aided by heat, acts upon metallic tin, with liberation of sulphurous anhydride and sulphur, and the eventual production of stannic sulphate. Tin is not attacked by the strongest nitric acid, but by acid of a density of 1·30 it is violently acted upon, with formation of a white, crystalloid, insoluble substance, metastannic acid, $5SnO_2.10H_2O$. When attacked in the cold by very weak nitric acid tin is slowly dissolved with formation of stannous nitrate, and nitrate of ammonium. Tin is oxidizable by fused caustic potash or soda, and more readily so by a fused mixture of an alkaline nitrate with an alkaline hydrate or carbonate. Strong hot solutions of the caustic alkalies react on metallic tin, with evolution of hydrogen and the production of an alkaline stannate.

On account of its general inalterability, this metal is extensively employed for domestic purposes. It is largely used for tinning the insides of copper vessels, for tin-foil, and for the manufacture of tin-plates; it is also an essential ingredient of pewter, plumbers' solder, bronze, and various other important alloys.

## TIN ORES.

CASSITERITE; OXIDE OF TIN; *Étain oxydé; Zinnstein.* Tetragonal.—The tin of commerce is obtained from the native oxide of that metal, which belongs chiefly to the older formations, and is usually met with in veins traversing granite, gneiss, or mica-slate. This mineral has, however, been recently found in limestone of Lower Liassic age near Campiglia Marittima, Tuscany, where it has evidently been extensively worked by the ancients. Oxide of tin has a specific gravity varying from 6·3 to 7·1. Its colour is usually brown or black, but sometimes red, grey, white, or yellow. It has an imperfect fracture, a grey streak, and a highly adamantine lustre. When pure this mineral consists of tin 78·62, oxygen 21·38. It is, however, frequently associated with other metals, particularly with arsenic and iron.

A specimen of cassiterite from Cornwall, analysed by Klaproth, gave the following results:—

| | |
|---|---|
| Sn | 77·50 |
| O | 21·50 |
| $Fe_2O_3$ | 0·25 |
| $SiO_2$ | 0·75 |
| | 100·00 |

The composition of this mineral is expressed by the formula $SnO_2$. The most commonly occurring crystals are rectangular prisms terminated by four triangular planes, which may be more or less modified on their edges and angles. It also occurs in botryoidal and reniform shapes, which are known as wood-tins.

Oxide of tin is infusible when heated alone before the blowpipe, and is not readily reduced to the metallic state without the aid of fluxes. It is insoluble in acids; but when heated on a charcoal support, with the addition of carbonate of sodium, it readily affords minute globules of metal.

Cornwall is one of the localities most productive of this mineral, where it occurs associated with copper and iron pyrites, wolfram, mica, and tourmaline, together with axinite, and other silicates. Tin mines are worked in Saxony, Austria, and Bohemia; in Peru and Bolivia; in

China, Malacca, and especially on the islands of Banca and Billiton, in the Indian Archipelago, whence large quantities are imported into this country. Queensland, New South Wales, and Tasmania annually yield large quantities of tin ore. This ore also occurs in Galicia, Spain; in Sweden; in the Department of Morbihan in France; in Greenland, Russia, Brazil, Mexico, Chili, and the United States of America.

TIN PYRITES; *Étain sulfuré; Zinnkies.* Cubic, tetrahedral.—This mineral has hitherto been found in a crystalline state only in Huel Rock Mine, in the parish of St. Agnes, Cornwall. Tin pyrites is of a yellowish-grey colour, and has a strong metallic lustre; it commonly occurs in granular amorphous masses, and has a specific gravity of 4·35. It affords a black streak, and presents an uneven fracture. When heated before the blowpipe, sulphide of tin fuses into a black slag, which is extremely difficult of reduction. It is attacked by nitric acid, and affords on subsidence an abundant white precipitate of metastannic acid. Two analyses of tin pyrites yielded the following results:—

| | Kudernatsch. | Klaproth. |
|---|---|---|
| Sn . . . . | 25·55 | 26·5 |
| Cu . . . . | 29·39 | 30·0 |
| Fe . . . . | 12·44 | 12·0 |
| S . . . . | 29·64 | 30·5 |
| | 97·02 | 99·0 |

This mineral does not occur in sufficient quantities to admit of being metallurgically treated, and must be regarded as essentially a mixture of various isomorphous sulphides.

## DISTRIBUTION OF TIN ORES.

Tin has, more than almost any other metal, a characteristic mode of occurrence, being usually found in the older crystalline and metamorphic rocks.

The only ore yielding tin in commercial quantities is cassiterite, which occurs in four different forms of deposit. Firstly, in *veins* or lodes, from which the larger proportion of the tin annually produced in this country is obtained. Secondly, in *flats*, usually connected with true veins, but passing into the enclosing rocks, and sometimes forming deposits parallel to their stratification. Thirdly, in *stockworks*, which chiefly occur in granite, and consist of numerous minute veins or impregnations of cassiterite passing through the rock in all directions. Fourthly, as *stream tin*, which consists of water-worn nodules and grains of tin oxide occurring in alluvial sands and gravels. This variety of tin ore is obtained by a process of washing very similar to that employed for the separation of alluvial gold. Although this metal has been in use from remote antiquity, up to a comparatively recent date Cornwall produced a very large proportion of that annually brought into the market. About the year 1710 rich deposits of tin ore were discovered in the island of Banca in the Malay Archipelago, and, more recently, in the neighbouring island of Billiton, in the Straits Settlements, and in various parts of the Australian Colonies.

In Cornwall, where tin ores have been continuously raised from the time of the Phœnicians, the deposits comprehend all the different varieties previously enumerated; those of stream tin have, however, become almost entirely exhausted. Tin veins usually occur either in granite or in *killas* or clay-slate, and are generally most productive in the vicinity of the junction of these rocks. The gangue of stanniferous deposits generally consists of quartzose matter, and the minerals associated with cassiterite are remarkably constant, consisting of wolfram, pyrites, mispickel, apatite, topaz, mica, tourmaline, &c.

During the years 1835 and 1838 the annual production of tin in Cornwall and Devon amounted to between 4,000 and 5,000 tons; afterwards it steadily increased, until in 1871 it amounted to 16,898 tons of *black tin*,[1] equivalent to 11,320 tons of metal. In 1883 the production of black tin in Cornwall and Devon amounted to 14,468 tons, equivalent to 9,307 tons of metallic tin.

In Saxony a small quantity of tin is annually raised, but the amount is insignificant when compared with that produced in England. The chief Saxon and Bohemian localities are Altenberg, Geyer, and Zinnwald, in the Erzgebirge, and Schlackenwald near Elbogen.

At Altenberg true stockwork-deposits occur, consisting of a granitic rock, intersected by numerous small interlacing veins of tinstone. The value of the tin raised at Altenberg in 1880 amounted to £9,105.

At Geyer the rock in which the ore occurs is a granite, consisting largely of decomposed felspar, and containing apatite, tourmaline, and fluor-spar. The tinstone is in small parallel veins and is disseminated through the rock; the veins, which are rarely more than 2 inches in width, merge into the enclosing rock without exhibiting any distinct walls.

At Zinnwald the tin occurs in masses of granite frequently enclosed in porphyry, the most productive deposits being composed of quartz and cassiterite in thin nearly horizontal layers; in some cases, however, the whole rock is stanniferous.

From France and Spain tin is to a great extent absent, rarely occurring in workable quantities. A few tons are, however, annually produced in both countries, the ore being obtained from Brittany, and from various localities in Galicia.

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[1] Concentrated tin oxide prepared for smelting.

Another great centre of production includes Banca and Billiton or Blitong, in which all the ore worked is in the form of detrital or stream tin. Veins have, however, been found in considerable numbers, but not of sufficient size to pay for working. The alluvial deposits are in most cases covered by beds of variously coloured sands to a depth of from 10 to 15 feet, and rest on a stratum of white clay, which is considered an infallible indication of the limit of the stanniferous beds. It is stated that the stream tin of Banca is derived from the granite or rocks immediately contiguous to it, and that tin is only found in valleys the streams of which take their rise in these rocks. During the year 1882 Banca and Billiton, together, produced 8,550 tons of metallic tin.

In addition to the tin produced by these islands, there is also a large quantity annually raised on the Malay peninsula. In the year 1882 above 7,000 tons of metallic tin were exported from Perak.

During the last twelve years large quantities of tin ore, chiefly obtained from streamworks, have been produced in the various Australian Colonies. In 1882 Victoria produced 1,077 tons of tin ore, while in the same year New South Wales exported tin and tin ore to the value of £833,461, and Queensland to the value of £560,590. The exports of tin ore and ingots from Tasmania during the year 1881 represented a value of £375,775.

In the United States of America cassiterite has been found in several localities, but never in workable quantities. Small quantities have been raised at Temescal, near Los Angeles, in California, and at the Brewer Mine, South Carolina. Recently large deposits of tin ore are stated to have been discovered in the Black Hills of Dacota. In South America tin mines are worked in Bolivia and Peru. The annual production of this metal in South America is estimated at 1,000 tons.

Stream tin has been found in several localities in Mexico, notably in Guanaxuato, Zacatecas, and Durango, where it is associated with topaz.

The production of metallic tin over the whole world during the year 1885, including the small quantities yielded by Peru and Bolivia, is estimated at 50,000 tons.

## ASSAY OF TIN ORES.

PREPARATION OF THE ORE.—Before proceeding to the assay of any description of "tin-stuff"[1] it is necessary to first isolate the oxide of tin, black tin, from the siliceous gangue and the various sulphurous and arsenical ores with which it may be associated.

This can be accomplished, in a somewhat rough way, by treatment somewhat similar to that by which the concentration and purification of tin ores are conducted on the large scale. With this view the pulverized mineral may be first roasted, and afterwards washed in a bowl, or in some other convenient vessel, until the lighter substances with which it is associated have been removed. In the tin-mining districts of Cornwall and Devon a large round-pointed shovel is employed for this purpose, and after each successive washing the heavier portions, which have not been carried off in suspension, are further reduced in size by grinding under a heavy hammer, the faces of which are slightly rounded. The assay is subsequently roasted and again washed. Roasting has for its object the decomposition of arsenical pyrites and of various minerals containing sulphur, which are greatly reduced in density after calcination, and are then easily removed by water. In Saxony, a small hand shaking-table is employed for this purpose, but it possesses no advantage over the ordinary *vanning-shovel.* Instead, however, of resorting to washing and roasting, the removal of arsenical and sulphurous minerals may be more completely and expeditiously effected by boiling the pulverized material with excess of nitro-hydrochloric acid, by which arsenical and common iron pyrites, copper ores, &c., are completely dissolved. The insoluble matters remaining in the flask will chiefly consist of tin oxide and silica, with sometimes a certain amount of tungstic acid, which may be removed by digestion with ammonia. What now remains will be a mixture of tin oxide with silica, from which the latter may be removed either by careful vanning, or, still better, by digestion with hydrofluoric acid in a platinum dish. When arsenical pyrites or ordinary iron pyrites containing tin is to be assayed for that metal, it may be attacked by nitro-hydrochloric acid, washed by decantation, and treated with ammonia. It is then transferred to a platinum dish and digested with hydrofluoric acid; these operations must be repeated until pure cassiterite remains in the capsule. In the majority of cases it will be unnecessary to treat with ammonia after any but the first attack by aqua regia.

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[1] Any veinstone or other rock containing a workable amount of oxide of tin is called "tin-stuff."

Assay of Black Tin.—The nearly pure oxide, in the state in which it is delivered by the miner to the smelter, or after the gangue has been attacked first by aqua regia and subsequently by hydrofluoric acid, in the way described, may be reduced to the metallic state by various processes.

*In Brasqued or Black-Lead Crucibles.*—A weighed quantity of from 200 to 400 grains of the oxide may be placed either in a brasqued crucible carefully covered and luted with clay, or it may be mixed with one-fifth its weight of ground charcoal or anthracite, and introduced into an ordinary plumbago pot and placed in the assay-furnace.

During the first quarter of an hour the heat must be gradually raised to dull redness, after which it is elevated to a full bright redness, at which it should be kept for about ten minutes. The crucible and its contents are now carefully removed from the fire, without knocking, and allowed to cool, when the pot is broken, and the button of tin removed and weighed. In order to recover any particles of metal which may be disseminated through the brasque in the form of minute globules, it must be removed and carefully washed. The weight of the metal so obtained is added to that of the original button. In the same way, any unconsumed charcoal-powder or anthracite added to the assay in the black-lead crucible must be carefully vanned, and, should any metal adhere firmly to the sides of the pot, it must be removed, and its weight added to that of the original button.

*Cornish Method of Assay.*—In Cornwall, assays of black tin are usually conducted in a naked plumbago pot, which is first made red hot, and the assay, consisting of 2 ounces of washed ore mixed with a little anthracite, introduced. In case the assay should not fuse readily, a little fluor-spar is added, and after exposure during a quarter of an hour to a full red heat, the tin is rapidly poured into a small ingot-mould, and the slag examined for metal by pounding and washing.

*Fusion with Potassium Cyanide.*—The crucibles employed for this purpose, when 100 grains of black tin are operated on, should be of about 3 ounces capacity, and must be prepared by ramming into the bottom of each a layer, about half an inch in thickness, of commercial cyanide of potassium. The requisite amount of finely powdered tin ore is now intimately mixed with from four to five times its weight of potassium cyanide, and placed in a crucible prepared as previously directed. This, with its contents, is moderately heated in an assay-furnace, and, after having been kept for about ten minutes in a state of tranquil fusion, the pot is removed, gently tapped to facilitate the formation of a single button, and allowed to cool. By operating in this way fairly accurate results are obtained, and with ordinary care, a difference of more than ½ per cent. should not occur between two assays of the same ore.

In estimating the amount of tin present in pyrites, or in any other sulphurous or arsenical material, it would generally be inconvenient to operate on a quantity exceeding 400 grains. The amount of tin oxide obtained by treating such a quantity with nitro-hydrochloric acid, and subsequently with hydrofluoric acid, will, in many cases, not exceed a few grains, and a very small crucible must consequently be employed for its reduction by cyanide of potassium. This crucible may either be enclosed in a larger one and heated in the assay-furnace, or be placed naked in the muffle. On examining the crucible after complete fusion, the reduced tin will be found to have assumed the form of a button of almost silvery whiteness covered by a layer of transparent flux. In order to collect any traces of metal occurring in the form of minute shot, the flux must be dissolved in hot water, and any metallic globules which may be found weighed with the principal button.

## ROASTING TIN ORES.

Tin ores, after the most complete concentration which can be effected by washing only, are generally contaminated with variable quantities of arsenical and ordinary pyrites, &c.

ROASTING IN REVERBERATORY FURNACES.—For the removal of these impurities the ores are taken to the *burning-house*, where the sulphides and arsenides are decomposed by roasting in reverberatory furnaces. These are usually from 12 to 15 feet in length, and from 7 to 9 feet in width; the hearth is horizontal; and the arch, which is about 2 feet in height in the neighbourhood of the fire-bridge, sinks gradually towards the chimney. This arrangement is provided with but one opening, closed by an iron door, placed at the extremity furthest removed from the grate, and immediately under a brick hood, by which the sulphurous and arsenical fumes are carried directly off into the chimney without annoyance or injury to the workmen.

In connection with the flues of these furnaces are condensing-chambers, in which arsenious oxide is deposited in a crystalline form. This is subsequently purified by a second sublimation, in order to convert it into the white arsenic of commerce.

From 10 to 15 cwts. of ore constitute a charge for one of these furnaces, and this requires from twelve to eighteen hours, according to the amount of pyrites present, before it is sufficiently roasted. The charging is effected by a small hopper in the centre of the brick arch; and as soon as the proper quantity of ore has been introduced it is regularly spread over the bottom. At the commencement of the operation the heat is very gradually raised until it reaches dull redness, at which temperature it is afterwards kept during several successive hours. At intervals during the process of calcination the mineral is stirred with an iron rake, so as to expose new surfaces.

When the ore has been sufficiently roasted, which is indicated by its ceasing to evolve white fumes, an iron plate fitted into the floor of the furnace is removed, and the charge, while still hot, is raked through the aperture, and falls into an arched chamber beneath, where it is allowed to cool. The calcined ore is afterwards again subjected to the process of washing; and the various impurities, which have been decomposed and chiefly transformed into ferric oxide, are, from their reduced specific gravity, readily removed. When the ore is contaminated with copper pyrites, it is, after being carefully roasted, allowed to remain for some time exposed to the atmosphere previously to being again washed, as a portion of the sulphide of copper is thus oxidized and converted into sulphate, which, being soluble in water, is easily removed. If tin ores contain much copper, it is usual to treat them, after their removal from the burning-house, with dilute sulphuric acid, by which copper oxide is dissolved, while the oxide of tin remains unaffected. After this treatment with sulphuric acid, the ore is washed in pure water, and is now ready to be handed over to the smelter.

The old reverberatory furnace has been to a great extent superseded by Brunton's calciner, with a rotative hearth, in which manual labour for turning the ore is dispensed with.

THE OXLAND AND HOCKING CALCINER.—Brunton's rotating calciner has in its turn to a large extent been replaced by the calciner of Messrs. Oxland and Hocking, figs. 140, 141, which is especially adapted for the treatment of *rank ores* containing much arsenic and sulphur.

This apparatus consists of a fire-chamber, A, from which the heat and products of combustion pass through an iron cylinder, B, made of boiler-plates, lined with fire-bricks placed on edge. The cylinder, which is from 30 to 40 feet long, and from 4 to 6 feet in diameter, is supported in an inclined position, but varying in inclination according to the character of the ore treated. It is supported on three pairs of friction-wheels, C, and is rotated by gearing, D, generally driven by a turbine or water-wheel. At the lower end it passes into the fire-chamber, and is so arranged as to deliver the ore passing through it, by an opening, *e*, in the arch, into the ore-chamber, F. At the upper end it communicates with the flues or condensing-chambers, G. The ore, brought from the dressing-floors in a wet condition, is dried on the cast-iron plates, *g*, covering these chambers. It is fed into the hopper, *h*, by a boy, who also attends to the fire. The cylinder revolves at the rate of from three to eight revolutions per minute, the ore being raised by four projecting lines of bricks parallel with its axis, but leaving room for the continuous running-in of dry ore from the hopper. When the ore has been raised sufficiently high on one of these shelves it falls off in thin streams through the hot gases passing up the cylinder. It thus becomes sufficiently heated for the sulphur and arsenic to take fire, and to burn with such energy that before the ore arrives half-way down the cylinder the greater portion of the arsenic and much of the sulphur is driven off. The heat evolved by the combustion of arsenic and sulphur is thus rendered available for heating the upper portion of the tube. As the partially calcined ore passes onwards beyond this point, the remaining arsenic and sulphur are not in sufficient quantity to produce the temperature required for the completion of the calcination, and hence the provision of a small fire-grate, H, for supplying the deficiency. Throughout the whole length of the cylinder, the lines of shelf perform the duty of passing the ore in finely divided streams through the heated gases, in such a way that no particle can escape full exposure to the oxidizing influences required for calcination. It is found that the arsenic burns off first, and that its removal is completed some time before the last portions of the sulphur are eliminated. The calcined ore, passing from the lower end of the cylinder into the ore-chamber, at a red heat, contains only traces of arsenic and but a small proportion of sulphur. When required it is withdrawn from the chamber, F, through the doors, *f*. Heated air for the more perfect combustion of the gases from the fire-place, and for the oxidation of the sulphur and arsenic, is supplied through the channel, *i*, above the arch.

By carefully controlling this supply of air and by maintaining a small consumption of coal, economy of fuel is not only effected, but another important result is obtained, namely, the passing of a minimum quantity of air through the calciner. The arsenious anhydride produced is con-

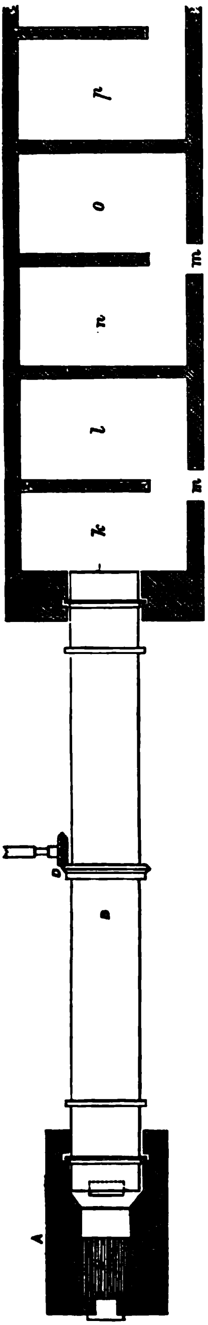

Fig. 140.—The Oxland and Hocking Calciner; plan, partly section.

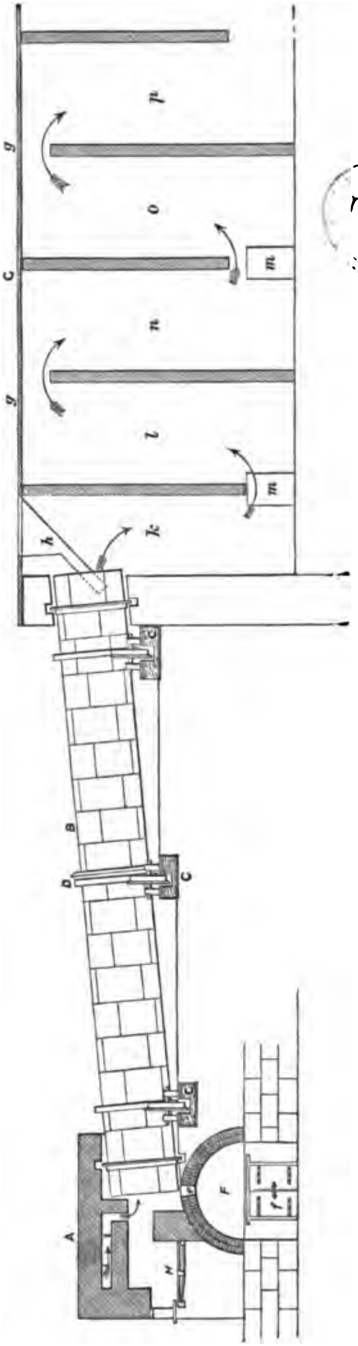

Fig. 141.—The Oxland and Hocking Calciner; elevation, partly section.

sequently condensed more easily than by the use of the reverberatory furnace or of Brunton's calciner, and perfect condensation is effected with less costly condensing-chambers.

The consumption of fuel is small, and the condition of the calcined product is well suited for subsequent operations. By the rotation of the cylinder, the crown, after being fully heated, is brought down under the ore, becoming in its turn the bed, thus preventing the loss of heat necessarily incurred in other furnaces from the crown only being exposed to heat, while the bed is covered by the ore.

The condensation of the arsenious oxide is much promoted by covering the condensing chambers with cast-iron plates, so that the damp ores placed upon them to dry, may, by cooling them, cause a proportionately rapid condensation. Formerly the condensing flues and chambers were constructed with thick walls covered by stone arches; but at Devon Great Consols and at East Pool Mines, Cornwall, great advantage has been derived from building flues of thin brickwork, and covering them with cast-iron plates. The openings used for removing the arsenic are in like manner closed by iron plates. Any fine ore carried off by the draught is deposited in the divisions *k* and *l* of the condenser, and is removed from time to time by the first door, *m*. The arsenious oxide is deposited in the compartments *n*, *o*, *p*, &c., and is removed through other doors, while the sulphurous anhydride passes on by the flue to the chimney.

Large quantities of crude arsenious oxide are produced in the mines of Devon and Cornwall. At New Great Consols, near Tavistock, nearly 10 tons of arsenious oxide were, at one time, obtained in the production of 1 ton of marketable tin ore, while at East Pool 2 tons were sometimes produced in dressing each ton of black tin. This *crude arsenic*, of a grey colour, is sold to the arsenic manufacturers, by whom it is refined before it is finally sent to the market. Large quantities of arsenious oxide are annually produced in Cornwall and Devon from the calcination of ores of tin and copper. In the year 1881 Great Devon Consols alone produced 2,851 tons of white arsenic, value £23,324, from the treatment of low-grade copper ores.

The calcined product from tin mines consists principally of oxides of iron and copper, containing sulphur, arsenic, and occasionally traces of cobalt and nickel, and in some localities a large proportion of wolfram, with silica and alumina; oxide of tin may be present in quantities varying from 2 to 20 per cent.

The ore is removed from the receiving-chamber and cooled by moistening with water; it is now known as *burnt witts*, and is taken to the burning-house floors, where, by treatment in circular buddles, the tin is concentrated by the removal of the oxides of iron and earthy matters, until it contains from 50 to 60 per cent. of black tin. It is then again calcined for the separation of the last traces of sulphur and arsenic, again passed through the buddles, washed in *kieves* or tubs, and finally concentrated until it will produce by assay from 65 to 70 per cent. of metallic tin.

SEPARATION OF TUNGSTEN; OXLAND'S PROCESS.—If wolfram be present in large quantities, the black tin, with which it is associated, is much reduced in value. This mineral cannot be separated from tin ore by any process of washing, since their specific gravity is nearly the same. The calcining operations already described have no effect on this substance; but a process introduced by Mr. R. Oxland, first at Drake Walls, and subsequently at East Pool, is said to effect a perfect separation.

This process consists in the conversion of wolfram (tungstate of iron and manganese) into tungstate of sodium, which being readily soluble in water is thereby removed, leaving the oxides of iron and manganese in a finely divided state, and so light as to be readily separated from black tin by washing. A reverberatory furnace is used, of which the peculiar characteristic consists in the employment of a cast-iron bed. The charge is introduced into this furnace through a hopper in the crown of the arch and is spread upon the iron bed, and exposed to the flame passing from the fire-place through the body of the furnace. The products of combustion then pass down to a flue formed by a diagonal partition of brickwork, which conducts it to the front of the furnace under the iron bottom, and then, returning on the other side, to the chimney. Thus, the whole of the bed is enveloped in heated gases passing off from the fire-place. The charge consists of from 10 to 11 cwts., according to the quality of the ore, and is prepared by mixing soda-ash with the dry ore in such proportions as to contain so much sodium as to be slightly in excess of that necessary to combine with tungstic acid. If the mixture be comparatively coarse-grained, a larger quantity can be operated on at a time than when it is in the state of slime. The charge is carefully turned over, so as to bring the whole to a bright red heat. It is known to be working well when it frizzles, becomes apparently moist, and is slightly adhesive to the tools used in stirring. If well worked, the charge should be drawn in from two and half to three hours, and is then in better condition than if exposed to a stronger heat or for a longer time. It is drawn in successive quantities, as soon as it is ready, through an aperture in the hearth, into a vault beneath, whence it is taken to the lixiviating-vats, where, by treatment with water while still hot, the tungstate of sodium is dissolved and run off into receivers. By successive affusions of water the whole of the saline matters are obtained as a clear liquid, the mass of the ore serving as a filter. The strong solutions thus prepared are either set aside to crystallize, or are evaporated to dryness in iron pans, affording crude tungstate of sodium, containing about 70 per cent. of the dry salt.

After lixiviation has been completed, the residue in the vats is conveyed to the dressing-floors, where, by washing with water, the oxides of manganese and iron, the residual constituents of wolfram, are carried off in suspension, making the water thick and of a reddish-brown colour. This process was for many years in operation at Drake Walls Mine, but the wolfram having subsequently disappeared from the lode there was no longer any necessity for its use. Salt-cake or crude sulphate of sodium may be substituted for the more expensive soda-ash, but as it requires skilful management it is not generally used.

Tungstate of sodium commands a limited sale for dyeing, for the manufacture of non-inflammable starch, for the production of a bronze-powder, and for some other purposes.

## METALLURGY OF TIN.

Tin-smelting in this country is now invariably conducted in reverberatory furnaces, although blast-furnaces were formerly employed for the purpose, and old smelting works, known as Blowing-Houses, where this method of treatment was at one time followed, are still to be found in a dismantled state in various parts of Cornwall.

**Treatment of Tin Ore (Black Tin) in the Reverberatory Furnace.**—The treatment of tin ores in the reverberatory furnace comprehends three distinct operations, namely, *smelting*, *refining*, and *re-melting the slags and residues.* These operations are conducted in a furnace of which fig. 142 is a longitudinal and fig. 143 a horizontal section, a little above the level of the hearth. The form and dimensions of these furnaces vary somewhat in different establishments, but the length of the hearth in that given as an illustration, which is now in daily use, is 12 feet, and its greatest width 8 feet.

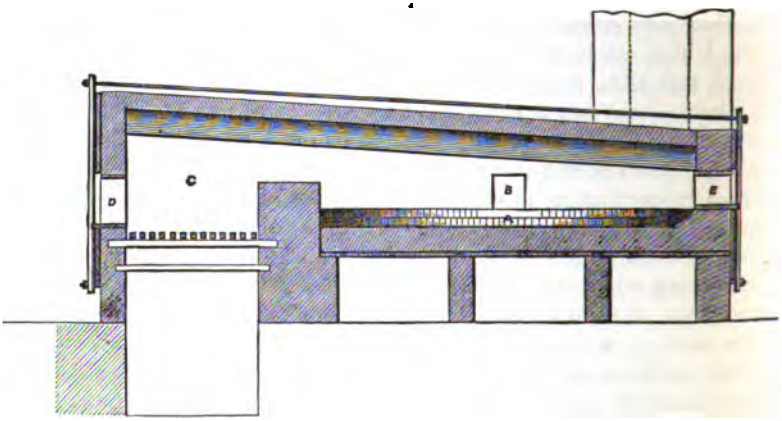

Fig. 142.—Tin Furnace; longitudinal section.

*Smelting.*—The ore, which usually contains from 65 to 70 per cent. of metallic tin, is mixed with one-fifth of its weight of small anthracite, *culm*, and is slightly sprinkled with water, for the double purpose of rendering it more easy to charge, and also to prevent any portion from being carried off mechanically by the draught. The charge of about 22 cwts. is thrown upon the bottom, A, through the door B, and the heat of the furnace is maintained by the fire-place, C, which is supplied with coal through the door, D. The charge is subsequently spread by means of tools introduced through the door, E. Each furnace is usually provided with a separate chimney, with which it is connected by a diagonal flue, not shown in the drawings. The temperature is gradually increased during the first five hours, at the expiration of which time the charge is well worked up with a rabble ; this stirring is repeated at the end of five hours and three quarters, and in about six hours from the time of charging, the tapping usually takes place. Two products are thus obtained, namely, metal and slag. The tin and slag are run off through the tap-hole, *f*, into the float, G, which is lined with fire-clay, and has usually a rectangular form with rounded corners.

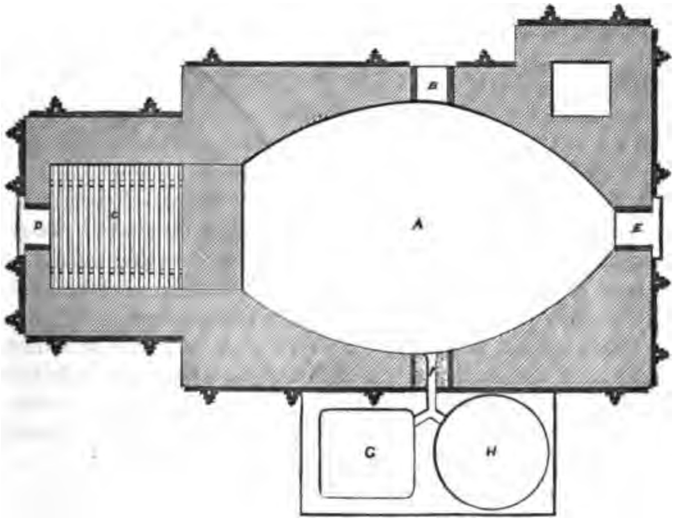

Fig. 143.—Tin Furnace ; horizontal section.

The slag, when sufficiently chilled, is removed from the surface of the tin in the float, which is skimmed and laded into moulds which give it the form of blocks.

The spongy slags remaining on the bottom of the furnace containing globules of metallic tin and frequently pieces of unconsumed culm, or culm ashes, are drawn through the door, E, and reserved for subsequent treatment.

*Refining.*—This process comprehends two distinct operations, *liquation* and *poling* or *tossing*. Blocks of metallic tin, to the amount of about 6 tons, resulting from the treatment of black tin, are arranged on the hearth of the reverberatory furnace employed for smelting, and the temperature is slowly raised. By this treatment the more readily fusible tin is eliquated, and flowing over the surface of the hearth, and escaping by the tap-hole, *f*, is collected in the cast-iron "kettle," H, which is set over a small independent fire-place. In proportion as the blocks are gradually eliquated and their bulk thereby becomes reduced, other blocks of tin are introduced into the furnace. At the last, when metallic tin has ceased to flow at a moderate heat, the temperature is considerably increased, and the more refractory material left on the furnace bottom is melted and runs out into the float. The metal thus obtained, which, in addition to tin, contains small quantities of arsenic, sulphur, and iron, is laded into moulds and laid aside to be subsequently again treated by liquation. The process of liquation is usually carried on at the end of each week, after allowing the temperature of the furnace to become sufficiently reduced.

The charge of the kettle, H, consists of about 7 tons of eliquated tin, which is kept hot by the fire beneath it. This is poled by forcing into it, by a crutch, supported by an iron jib, a piece of green wood, that of the apple-tree being generally preferred.

When tin of common or second quality is being prepared, this boiling is not long continued, and the dross which has risen to the surface is skimmed off and thrown back into the furnace, after which the tin is laded into moulds. Tin still continues to flow into the kettle during the time the metal is being dipped out and cast into moulds. In making "refined tin" the bath of eliquated metal is poled during several hours.

Instead of boiling the metal by the introduction of billets of green wood, the same effect is sometimes produced by tossing. When this process is employed the agitation is produced by the workmen continually lifting the melted metal in a ladle, and letting it fall from a considerable height into the kettle. The scum thus brought to the surface is carefully removed by skimming, and the metal finally laded into moulds.

*Re-melting the Slags and Residues.*—The slags run off with the tin in the first operation, and withdrawn from the top of the float, are re-melted with lime, and sometimes with a little fluor-spar, in a separate but exactly similar furnace. The metal obtained by this operation, together with the resulting slags, is tapped into the float, and the tin is subsequently subjected to liquation. The slag separated from the metal is a black vitreous substance, closely resembling obsidian, and is practically free from tin. The slags drawn through the door, E, of the smelting furnace, at the termination of the first operation, are stamped and washed for the buttons (*prillons*) of metal which they contain. The comparatively infusible residues resulting from successive processes of liquation are, after fusion, again subjected to liquation at somewhat more elevated temperatures. The final residue remaining on the furnace bottom, is, after fusion, known as *hard-head*. A specimen of hard-head, from which as much as possible of the tin had been removed, afforded by analysis the following results:—

| | |
|---|---|
| Fe | 62·50 |
| Sn | 17·25 |
| As | 19·02 |
| S | 1·26 |
| | 100·03 |

Tin slags are essentially ferrous silicates, containing a little tin, lime, and tungsten.

The total consumption of fuel per ton of metallic tin produced is about 30 cwts.

The tin smelted in this country is usually cast into "blocks" of 4 cwts., "slabs" of 84 lbs., "ingots" of 56 and 28 lbs., and "bars" of 3 ounces.

Three distinct qualities of tin are recognised in the trade, namely, *common*, *refined*, and *grain tin*. Grain tin is prepared by heating blocks of the purest tin, and when the temperature has been sufficiently elevated to render the mass brittle and to cause the block to assume a crystalline structure, it is broken either by a fall or by a blow from a heavy mallet. Grain tin is largely employed in the preparation of tin salts.

**SMELTING IN THE BLAST-FURNACE.**—All the tin raised in Cornwall was formerly smelted in blast-furnaces, and this method of treatment was applied to stream tin long after the more impure mine tin was generally reduced in the reverberatory furnace.

Pryce in his *Mineralogia Cornubiensis*, published in 1778, states that in his time mine tin "corrupted with some portion of mundick and other minerals" was smelted with pit-coal in reverberatory furnaces. On the other hand, stream tin, free from such impurities, was smelted in blast-furnaces with charcoal, yielding a metal which from its great purity was worth £12 a ton more than ordinary mine tin. The following is his description of smelting in the blast-furnace as then practised:[1]—"The furnace itself for blowing the tin is called the castle on account of its strength, being of massive stones cramped together with iron to endure the united force of fire and air. This fire is made with charcoal excited by two large bellows, which are worked by a water-wheel, the same as at the iron forges. They are about 8 feet long and 2½ wide at the broadest part. The fire-place, or castle, is about 6 feet perpendicular, 2 feet wide in the top part, each way, and about 14 inches in the bottom. all made of moorstone and clay well cemented and cramped together, The pipe or nose of each bellows is fixed 10 inches high from the bottom of the castle in a large piece of wrought-iron, called the earth-eye. The tin and charcoal are laid in the castle stratum super stratum in such quantities as are thought proper; so that from 8 to 12 cwt. of tin, by the consumption of 18 to 24 sixty-gallon packs of charcoal, may be smelted in a tide or twelve hours time. These bellows are not only useful for igniting the charcoal, but they throw in a steady and powerful air into the castle, which, at the same time that it smelts the tin, forces it out also through a hole at the bottom of the castle, about 4 inches high and 1½ inch wide, into a moorstone trough 6½ feet high and 1 foot wide, called the float, whence it is laded into lesser troughs or moulds, each of which contains about 3 cwts. of metal called slabs, blocks or pieces of tin, in which size and form it is sold in every market in Europe."

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[1] 'Mineralogia Cornubiensis,' p. 136.

In the Erzgebirge, a blast-furnace about 10 feet in height is employed. The sides are formed of large pieces of granite, and the hearth is a block of the same material, lined with brasque, and having a considerable fall towards the breast. The fused matter escaping from this cavity flows continuously into an exterior basin lined with brasque. This is furnished with a tap-hole, by which its contents may be withdrawn into a small iron vessel heated by a separate fire and placed at a lower level. The charcoal and ore are introduced by successive charges, and the blast is furnished by a small blowing-machine. The slags produced float on the surface of the metal collected in the basin, whence they are removed by an iron hook as soon as they have sufficiently solidified. When the reservoir has in this way become filled with metal, the tapping-hole is opened and its contents are run into the iron vessel, where the process of refining is conducted by boiling with green wood and skimming. The slags are divided into two classes; the richer are, without any mechanical preparation, fused with succeeding charges of ore; the poorer, after being stamped, are washed for the purpose of separating the metallic granules which they contain. By this process every ton of tin produced requires for its reduction $1\frac{6}{10}$ ton of charcoal, and the loss of metal is somewhat greater than in the reverberatory furnace.

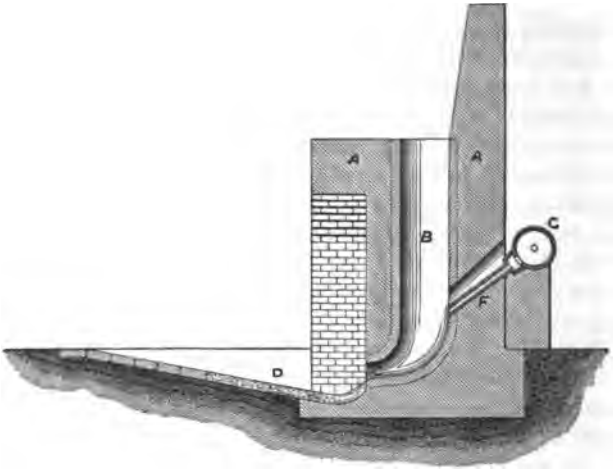

Fig. 144.—Tin Furnace, Perak ; vertical section.

Perak, which forms a portion of the Malayan Peninsula, including the district of Laroute, annually produces above 7,000 tons of metallic tin, exclusively derived from stream tin smelted in blast-furnaces.

The furnaces employed vary considerably in form and dimensions, but that represented in vertical section by fig. 144, and in horizontal section by fig. 145, is generally adopted in all the larger metallurgical establishments of that region.[1] It consists of brickwork, A, standing 6 feet above the level of the floor, and having a width of 7 feet 6 inches. This encloses a nearly semi-cylindrical cavity, B, 18 inches in diameter in its upper portion, but gradually diminishing downwards until it terminates at the breast of the furnace in an opening, C, only 6 inches in diameter. In front of this is an inclined float, D, the lower portion of which receives the fused matter issuing from the furnace, while the masonry at the back is raised 4 feet above its top, and thus serves as a support for the charges, which rest against it. The mouth of this furnace is reached by means of steps, E, arranged on either side of it, and at the back a tuyer, F, passes through an opening left for that purpose. This opening is sufficiently large to admit the passage of an iron rod, in case the bottom of the furnace should become clogged, but is usually stopped during its working by the introduction of a little moist clay.

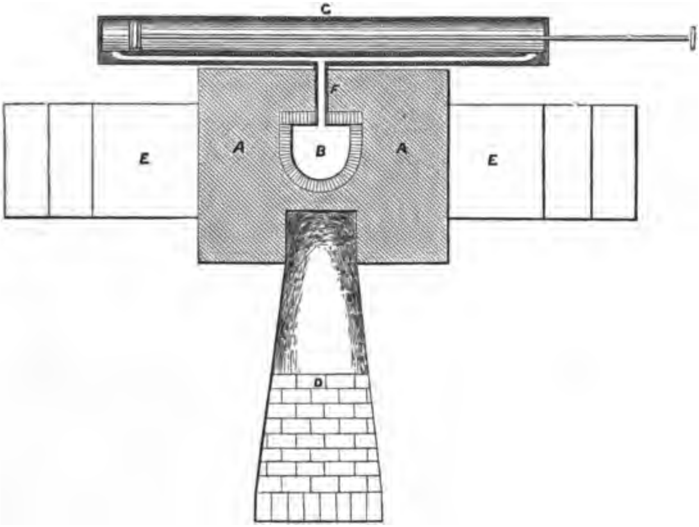

Fig. 145.—Tin Furnace, Perak; horizontal section.

The blowing apparatus, G, is placed at the back of the furnace, and consists of a wooden cylinder 13 feet in length and 16 inches in diameter, provided with a piston packed either with feathers or with paper. On the side of the cylinder next the back of the furnace is an air-way 4 inches in diameter, which at its two extremes communicates with the cylinder and in the middle with the tuyer. The wooden piston-rod, which is about 20 feet in length, is well greased, and passes through a closely fitting hole in the end of the cylinder. At each end of the cylinder is a valve 8 inches square opening inward, which alternately opens and shuts at each stroke of the piston. The tuyer, which is made either of iron or of burnt clay, is inclined at an angle of from 40° to 45°, and is directed towards the centre of the fore-breast. The blast is supplied by the labour of three men working together, who are relieved every hour, and in the majority of cases the furnace is only worked during twelve hours.

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[1] Les Mines d'Étain de Pérak par M. J. Errington de la Croix, p. 50. Paris, 1882.

The operation of smelting is commenced by filling the furnace with wood and charcoal, which is ignited and allowed to burn for about an hour; at the expiration of which time the cavity is completely filled with charcoal and the blast applied. A quarter of an hour after this the first ore, which is damped in order to prevent mechanical loss, is thrown upon the fuel, and, subsequently, charges of charcoal and mineral are alternately supplied.

The metal and slags run out of the furnace into the float, D, the slags as they cool being removed from the surface of the metal, which is from time to time laded into moulds composed of a mixture of clay and sand. The blocks of tin thus made usually weigh about 94 lbs., and the production of the shift of twelve hours is about 2,900 lbs., with an expenditure of an equal weight of charcoal.

The slags are ground in an iron mortar by a pestle worked by the foot, which in form closely resembles a tilt-hammer, and the beads of metal, separated by washing, are subsequently run down in a shallow wrought-iron pan provided with a fire-place, which is heated by wood. These granules after fusion are poured into moulds.

Only that portion of the Perak tin which is intended for the European markets is subjected to the operation of refining, which is carried on exclusively at Penang. In order to effect this object the blocks of unrefined tin are melted in a large cast-iron kettle, with a fire beneath it, and the fused metal is "tossed" with a ladle in the usual way. After a final skimming the metal is again cast into blocks.

In some of the smaller establishments on the Malayan Peninsula the furnace is made entirely of clay supported by wooden binders. Furnaces of this description often do not exceed a foot in internal diameter, and are usually about 3 feet 6 inches in height. The blast-furnace employed in Siam consists of a wrought-iron jacket lined with clay 3 feet in height; the internal diameter of this furnace is 18 inches. The method of procuring a blast is in all cases the same.

## ALLOYS OF TIN.

Tin is extensively employed for the manufacture of tin-plate, which is thin sheet-iron, externally coated with that metal, and for coating copper vessels. *Pewter* is an alloy of tin and lead, in the proportion of four parts of the former to one of the latter. *Solders* are also alloys of tin and lead in varying proportions; *fine solder* is composed of two parts of tin and one of lead; *tin solder*, used in the manufacture of articles of tin-plate, consists of a mixture of equal proportions of the two metals; *plumber's solder* is made by mixing together one part of tin with two of lead.

Large type is made of an alloy of lead and antimony only, but tin enters into the composition of the mixture used for some of the smaller descriptions. Tin alloyed with antimony constitutes *Britannia metal*, the best varieties being composed of tin with just sufficient antimony to give it hardness.

The addition of bismuth to alloys of tin and lead has the effect of lowering their melting-point to an extraordinary degree; an alloy of two parts of bismuth to one of lead and one of tin melts below 100° C.

Tin combines with copper in all proportions; ordinary *gun-metal* is a mixture of 10 per cent. of tin with 90 of copper. Genuine *bronze* is a compound of copper and tin only, but that used for statuary usually contains a certain proportion both of lead and zinc. *Bell-metal* is a bronze containing a very large proportion of tin. "Tom of Lincoln" is composed of 22 per cent. of tin and 78 of copper; "Big Ben" of about 24 per cent. of tin and 76 of copper.

## ANTIMONY.

Antimony is a brilliant metal, of a white colour, slightly inclining to blue. It fuses at a temperature just below redness, and contracts but slightly in becoming solid.

It is extremely brittle, and possesses a strongly crystalline texture, so that when broken it exhibits well-defined facets, indications of which may be observed on the surface of the cooled ingot. It is slowly but distinctly volatile at a white heat in closed vessels, but admits of being distilled in a current of hydrogen.

Antimony is not sensibly affected by exposure to air at ordinary temperatures, but is rapidly oxidized when exposed to it in a state of fusion. When fused and strongly heated antimony is allowed to fall from a certain height, combustion, accompanied by the production of a thick white smoke, immediately takes place. This white vapour is chiefly antimonious oxide, $Sb_2O_3$.

This metal does not occur in commerce in a state of purity, but is contaminated by variable quantities of iron, lead, arsenic, and sulphur. To separate these, it may, after being reduced to a fine powder in an iron mortar, be intimately mixed with one-tenth of its weight of nitre, and subsequently fused in an earthen crucible.

By this treatment, the impurities, together with a portion of the antimony, become oxidized, and on breaking the vessel, after having allowed it to cool, the antimony is obtained as a metallic button, the surface of which will be covered with a fern-like crystallization. The purification of this metal may likewise be effected by fusing it, when in a finely divided state, with a small quantity of antimonious oxide. Fineness of grain is an indication of the purity of metallic antimony.

When in a state of fine division, antimony is attacked by hydrochloric acid with the evolution of hydrogen gas, but is not acted on by dilute sulphuric acid. When attacked by hot concentrated sulphuric acid, it becomes oxidized with evolution of sulphurous anhydride. Nitric acid attacks antimony; the degree of oxidation varying with the strength of the acid. Aqua regia attacks antimony, and gives rise to the formation of antimonious chloride ($SbCl_3$), soluble in an excess of hydrochloric acid.

On electrolyzing a solution containing one part of tartar emetic and four parts of antimonious chloride, by a small battery of two elements, antimony forming the positive and copper the negative pole, metallic antimony is deposited which possesses the remarkable property of exploding and taking fire when scratched or broken.

## ANTIMONY ORES.

Antimony, although occasionally found in a native state, is usually combined with sulphur, and is often associated with galena. It also exists in combination with both oxygen and arsenic. Native antimony crystallizes in forms derived from the rhombohedron, and is often associated with small quantities of iron and silver.

STIBNITE; *Antimoine sulfuré; Grauspiessglaserz.* Rhombic.—This substance, which is almost the only mineral sufficiently abundant to be regarded, practically, as an ore of antimony, is of a lead or steel-grey colour, which is liable to tarnish on exposure.

The cleavage is parallel to the shorter diagonal, and the crystals are commonly divergent, columnar, or fibrous. It also occurs in granular amorphous masses. Its specific gravity varies from 4·62 to 4·70; its streak has the colour of the mineral itself, and on being heated on charcoal before the blowpipe, abundant white fumes and an odour of sulphur are evolved.

This ore, which is commonly associated with iron, zinc, lead, silver, quartz, and sulphate of barium, occurs in veins traversing granite, clay-slate, &c. Its most celebrated localities were formerly Felsőbánya and Schemnitz, in Hungary; Wolfsberg, in the Harz; and Puy-de-Dôme and Cantal, in France. Mines of sulphide of antimony have also been worked in Spain, Corsica, and in the County of Cornwall. Of late years the chief supply of antimonial ores has been obtained from Australia, Borneo, and Japan.

The analysis of a specimen of this mineral from Westphalia afforded Schnabel the following results:—

| | |
|---|---|
| Sb | 72·02 |
| S | 27·85 |
| Fe | ·13 |
| | 100·00 |

It is consequently a sulphide of antimony, of which the composition is represented by the formula $Sb_2S_3$.

VALENTINITE; Antimonious oxide, $Sb_2O_3$, occurs, though rarely, as a mineral, in shining white crystals belonging to the rhombic system. It is found in veins at Przibram, in Bohemia; at Braünsdorf, in Saxony; and at Malaczka, in Hungary. Antimonious oxide also occurs in regular octahedra as *senarmontite*, in the province of Constantine, Algeria; it is therefore dimorphous. Tetroxide of antimony, sometimes called *antimonious acid*, $Sb_2O_4$, is found native as *cervantite*, or *antimony ochre*, at Pereta, in Tuscany, and elsewhere. Antimony also occurs as *red antimony*, *antimony blende*, or *kermesite*, $Sb_2O_3.2Sb_2S_3$; likewise as sulphide combined with other metallic sulphides, particularly with those of lead and silver.

We have been unable to obtain any reliable statistics with regard to the annual production of antimony, but the amount raised in this country is exceedingly small. The imports of antimonial ores into the United Kingdom are now chiefly from Borneo and Japan.

## ASSAY OF ANTIMONY ORES.

DRY ASSAY.—From the great volatility of this metal its ores must be reduced at a moderate heat. For the purpose of assay, ores of antimony may be divided into two classes: the first comprehends all compounds in which the metal is either native or combined with oxygen, and in which little or no sulphur is present; the second consists of sulphide of antimony, and all other antimonial compounds containing large quantities of sulphur.

*Class* 1.—All substances belonging to this division are, when free from earthy or siliceous impurities, readily reduced by being moderately heated with finely divided charcoal. Their assay may be conducted in an earthen crucible lined with charcoal, without the addition of any flux.

As the volatility of this metal renders it necessary to avoid the application of a strong heat, when the ore to be examined is contaminated with siliceous impurities the addition of a suitable flux becomes necessary. For this purpose the ore may be either intimately mixed with two parts of black flux, or with one part of carbonate of sodium, and 0·25 of finely powdered charcoal. In this case, lining the crucible is unnecessary, and after it has remained in the fire until its contents are in a state of tranquil fusion, it should, on being withdrawn, be gently tapped against some hard body, to collect the fused metal into a compact button. When the crucible has become cold it is broken, and the button extracted and weighed. Care is, however, necessary in detaching it from adhering slag, since, from its brittleness, it is liable to become broken, and a portion consequently lost.

This method is likewise applicable to substances which, although principally composed of oxides of antimony, nevertheless contain small quantities of sulphur; as the sulphide yields with black flux just one-half of its antimony, a very small portion only can, in such cases, be retained in the slags. When oxide of iron is present in the substance treated, that metal is reduced at the same time as the antimony, and uniting with it forms an alloy, by which the result is to some extent vitiated.

*Class* 2.—The assay of substances belonging to this class may either be made by first roasting the sulphide, and subsequently fusing the oxidized residue with black flux, or by directly fusing the crude mineral with the addition of black flux and metallic iron or with iron scale. The roasting of sulphide of antimony is, from its fusibility and the facility with which it is sublimed, an operation requiring much care, and it must consequently be conducted at a very low heat, and be constantly kept stirred with an iron rod, until all smell of sulphur ceases to be evolved. The residue is then fused with three parts of black flux, and a button of antimony is obtained, as in the treatment of oxidized minerals belonging to the first class.

The antimony contained in the sulphide of that metal is readily liberated by fusion with metallic iron in a state of fine division. The sulphide of iron thus produced has, however, so very nearly the same density as metallic antimony, that their separation can only be obtained by keeping the contents of the crucible for a considerable time in a state of fusion. When this precaution is taken, two distinct buttons are obtained on breaking the crucible; the one, which is at the bottom, is of a white colour, and crystallized in large plates, whilst the other is of a bronze-yellow tint, and consists of sulphide of iron containing slight traces of antimony. These are carefully detached from one another, and the button of antimony weighed. The long-continued heat necessary to produce this separation has, however, the effect of causing the loss of a notable amount of antimony by sublimation, which is an inconvenience that cannot be entirely obviated by the most careful manipulation.

In operating in this way, it is of importance that only the amount of iron necessary to combine with the sulphur present should be added to the pounded sulphide, as any excess of that metal would combine with antimony, giving rise to an antimonide, which would contaminate the reduced metal.

For the reduction of pure sulphide of antimony, 42 per cent. of iron in the form of filings is required; these should be free from rust, and in the finest possible state of division, as when larger pieces are employed a considerable amount of antimony is lost by volatilization before they can be fully acted upon by the sulphide. Cast-iron must not be employed for the reduction of sulphide of antimony, as it is not only less readily acted on by sulphur than is wrought-iron, but the slag produced adheres so firmly to the reduced metal as to be difficult of removal.

If, instead of employing iron and sulphide of antimony alone, a certain proportion of carbonate of sodium and charcoal be added to the contents of the crucible, similar results are obtained, and a slag is produced containing sulphide of iron and sulphide of sodium.

A good mixture for this purpose consists of 100 parts of sulphide of antimony, 42 of metallic iron, 45 of carbonate of sodium, and 5 of finely powdered charcoal. When thus treated in a lined crucible, and at a moderate temperature, pure sulphide of antimony affords from 65 to 67 per cent. of metal.

Mitchell states that the best method of assaying sulphide of antimony is to mix it intimately with four parts of cyanide of potassium, and to heat it gently in an earthen crucible. The heat required in this case is so extremely low, that little if any of the metal is lost by sublimation; while by all other processes, a notable quantity, often amounting to 5 or 6 per cent., is driven off. It is consequently evident that the dry assay of antimony must rather be considered as a rough commercial approximation than as being exact. When pure sulphide is operated on its examination will be of but little value, since every 100 parts of that mineral correspond to 71·77 of antimony.

Wet Assay.—The volatility of antimony and of its sulphide is so great as to render all the various methods of estimating this metal by fire assay unsatisfactory. Even when the antimony is present in the form of oxide the following process will, however, afford accurate results.

The ore, after being pulverized and sifted in the usual way, is intimately incorporated with from 4 to 5 times its weight of a mixture consisting of equal parts of sulphur and sodium carbonate, placed in a covered porcelain crucible, and heated over a gas-burner. The mixture is thus thoroughly fused, and a portion of the sulphur driven off, while any oxide of antimony present is converted into sulphide of antimony. This sulphide is dissolved in the alkaline sulphide formed, and leaves the earthy gangue unattacked.

The crucible and its contents is now, after cooling, digested in hot water, until all the substances soluble in that menstruum have been dissolved, when the solution is passed through a filter. To the filtrate hydrochloric acid is carefully added until it becomes distinctly acid, and the whole of the sulphide of antimony, together with a considerable excess of sulphur, has been deposited.

This deposit is thrown upon a weighed filter, washed with water containing a little sulphuretted hydrogen, dried at 100° C. until its weight becomes constant, and the weight of the filter and contents noted. A weighed portion of the dried precipitate is now transferred to a small porcelain boat, and heated first gently, and then to a temperature of 200° to 230° C., in a piece of combustion-tubing about 16 inches in length, through which a current of carbonic anhydride is continuously passed. By this means the excess of sulphur will be volatilized and carried away, and after allowing the tube to cool, while the current of gas is still passing through it, the contents of the boat are weighed as antimonious sulphide, containing 71·77 per cent. of metallic antimony. From the weight thus found is calculated the percentage of antimony contained in the ore; but for the sake of verifying the result, it is desirable to operate in the same manner upon another weighed quantity of the dried precipitate obtained by the addition of hydrochloric acid to the original solution.

When an ore contains antimony sulphide, free from any admixture of oxide, it may be dissolved in strong hydrochloric acid, to which a little nitric acid has been added. Dilute slightly, and add ammonia and yellow sulphide of ammonium; boil, filter, and decompose the solution by adding slight excess of hydrochloric acid. The precipitated antimonious sulphide is collected on a filter, washed, dried, and heated in an atmosphere of carbonic anhydride. This is a quicker process than that by the mixture of sulphur when the ore contains no oxide of antimony insoluble in hydrochloric acid.

## METALLURGY OF ANTIMONY.

LIQUATION OF THE SULPHIDE.—From the fusibility of sulphide of antimony, its separation from the siliceous and earthy gangues with which it is associated is often effected by a simple liquation conducted at a moderate heat. On the Continent this operation is often carried on in vertical retorts; but in this country a reverberatory furnace has usually been employed.

In France, the separation of sulphide of antimony from its associated gangues is sometimes effected by means of an apparatus of which fig. 146 represents a vertical section. For this purpose the mineral is placed in large earthen retorts, R, of which four or more are set in each furnace. An aperture is left at the bottom of each, which corresponds with a similar opening in the tile, by which they are supported and to which they are luted. Beneath these, in separate chambers, C, are the earthen pots, P, in which is received the melted sulphide as it descends from the cylinders above.

The fuel consumed is wood, and the sulphide obtained is converted into metallic antimony, by roasting in a reverberatory furnace, and subsequent reduction by a mixture of 20 per cent. of pulverized charcoal, saturated with a strong solution of sodium carbonate.

REDUCTION TO THE METALLIC STATE IN CRUCIBLES.—To obtain metallic antimony, the sulphide is sometimes roasted in a reverberatory furnace until the sulphur has been expelled, and a grey residue remains. This is afterwards mixed with one-tenth its weight of crude tartar, and reduced in earthen crucibles. The metal obtained by this process is, with the exception of a certain admixture of iron, tolerably pure, and is ready for the market. The English process for antimony-smelting is conducted in large crucibles made of refractory clay mixed with a small proportion of plumbago, which are heated in circular wind-furnaces.

In order to obtain metallic antimony from its sulphide, by the English process, three distinct operations are required, namely, *Singling*, *Doubling*, and *Melting for star-metal.*

The furnaces used in this process are 3 feet in depth and 14 inches in diameter. The crucibles are 15 inches in depth, $10\frac{1}{2}$ inches wide at top, and 9 inches at bottom, inside measure; the fuel used is coke.

*Singling.*—This consists in fusing 40 lbs. of raw ore with from 20 to 22 lbs. of tin-plate clippings, by which treatment two products, sulphide of iron and impure metallic antimony, are obtained. In some cases a small quantity of slag from the next operation is added. Each fusion requires about $1\frac{1}{2}$ hour, and at its termination the charge is poured into a conical mould, and, when sufficiently cold, the antimony is separated from the ferruginous matte by which it is covered.

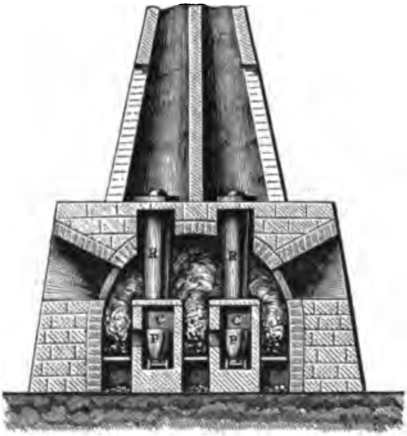

Fig. 146.—Liquation-Furnace ; vertical section.

*Doubling.*—The impure metal from the first operation is subsequently fused in a similar crucible with the addition of sodium sulphate and a small quantity of slag from the third process. The charge of each crucible is 80 lbs. of crude antimony, 2 lbs. of salt-cake, and a small quantity of slag from the star-metal. This fusion occupies 1 hour and 20 minutes.

*Melting for star-metal.*—About 60 lbs. of the metal obtained from the doubling process (*bowl-metal*) are broken into small fragments, to which are added 2 lbs. of pearlash and 5 lbs. of the slags obtained during a previous fusion for *star-* or *French-metal.* The fusion usually occupies somewhat less than one hour, and when it is completed the metal is cast into rectangular ingots, care being taken that each shall be completely covered with slag. If this be not attended to the necessary crystalline surface is not obtained.

Smelting in the Blast-Furnace.—In Bohemia antimony ores, consisting of sulphides and oxides, mixed with a considerable proportion of siliceous waste, are smelted in a blast-furnace for crude antimony, which is subsequently refined in a reverberatory furnace.

The average composition of the materials forming the blast-furnace charge is as follows:[1]—

| | Dressed Ores. | | Oxidized Ores. | Liquation Residues. | | Flue-Stuff. | Refining-Slag. |
|---|---|---|---|---|---|---|---|
| | Raw. | Calcined. | | Raw. | Calcined. | | |
| Sb . . | 43·3 | 48·9 | 46·4 | 21·4 | 23·1 | 56·1 | 25·7 |
| Fe . . | 14·4 | ... | ... | 12·9 | ... | ... | ... |
| S . . | 25·3 | 0·7 | 3·6 | 15·1 | 2·4 | ... | ... |
| $Fe_2O_3$ . | ... | 23·4 | 2·4 | ... | 17·3 | 6·9 | ... |
| $SiO_2$. . | 11·7 | 23·3 | 26·6 | 41·6 | 49·2 | 10·8 | 5·9 |
| $Al_2O_3$ . | 0·3 | 4·0 | ... | 0·5 | 2·4 | ... | ... |
| $CaCO_3$ . | 3·3 | ... | 7·0 | 4·0 | ... | ... | ... |
| CaO. . | ... | 1·0 | ... | 4·0 | 0·7 | ... | ... |
| $Sb_2S_3$ . | ... | ... | ... | ... | ... | ... | 2·4 |
| FeS. . | ... | ... | ... | ... | ... | ... | 53·4 |
| $Na_2S$ . | ... | ... | ... | ... | ... | ... | 9·0 |
| C . . | ... | ... | ... | ... | ... | 5·0 | ... |

The furnace has a round stack 6 metres in height, 1·4 metre in diameter at the throat, and 1 metre at the hearth, which is provided with five water-tuyers and three outlets for the molten products—an upper one for the slag, and two at the bottom. Of the latter, one has a slight rise, and forms the ordinary tap-hole for the metal, while the other, which is horizontal, is only used when blowing out the furnace.

The blast, supplied by a screw-blower, amounts to 15 cubic metres per minute at 30 c.m. water-pressure. The gases are collected by a tube leading from the side of the furnace a short distance below the throat. The furnace is kept continuously in blast for three weeks, and smelts about 19 metric tons of materials daily.

The smelting-mixtures are of two kinds, whose compositions are as follows:—

| | A.<br>Kilos. | B.<br>Kilos. |
|---|---|---|
| Calcined dressed ores . . . . . . . | 550 | 600 |
| Calcined liquation-residues . . . . . . | 750 | 600 |
| Ores balled with lime . . . . . . . | 200 | ... |
| Flue-stuff balled with lime . . . . . . | 100 | ... |
| Raw ore . . . . . . . . . . | ... | 100 |
| Oxidized ore . . . . . . . . . | ... | 100 |
| Raw liquation-residues . . . . . . . | ... | 100 |
| Limestone . . . . . . . . . . | 600 | 800 |
| Foul furnace-slags . . . . . . . | 400 | 400 |
| Refining-slags . . . . . . . . . | ... | 200 |
| Crude antimony, Metal No. III. . . . . . | ... | 100 |

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[1] 'Stahl und Eisen,' vol. vi., 1886, p. 62.

The most favourable result is obtained when these mixtures are used alternately, two charges of A being followed by one of B.

Lime is used to the extent of 10 per cent. to ball up part of the washed ore to a compact mass, and some of the flue-stuff is similarly treated, but with 7 per cent. of lime. This is, however, only necessary at starting, as it is found that when the furnace has been blowing for eight or ten days the materials may be charged in powder without in any way injuring its working.

The products of the blast-furnace are crude metallic antimony, slag, and flue-stuff. The first is divided into three classes, of the following average composition:—

| | No. I. | No. II. | No. III. |
|---|---|---|---|
| Sb | 90·02 | 73·80 | 65·04 |
| Fe | 6·23 | 16·66 | 23·80 |
| S | 2·85 | 8·42 | 10·46 |

The first two qualities are sent to the refinery, while No. III. goes back to the blast-furnace. The relative proportions are: No. I., 82·5; No. II., 9·0; and No. III., 8·5 per cent. of the total production of the furnace.

The slags produced vary slightly in accordance with the smelting-mixtures. Thus—

| | | |
|---|---|---|
| $SiO_2$ | 46·9 | 45·9 |
| CaO | 34·6 | 31·4 |
| FeO | 15·1 | 19·9 |
| Sb | 0·5 | 0·9 |

The first is produced from the mixture A, and the second from B.

There is very little flue-dust produced in the blast-furnace, the bulk of that product being derived from the preliminary operation of calcining the dressed ores. This is done in a single-bedded reverberatory furnace 8 metres long and 2 metres broad, with five working-doors on one side, heated with brown coal, which is burned upon a step grate. The produce is about 1,200 kilos. per day. The ore, which is charged in quantities of 200 kilos. at a time, clots and softens, giving off a large quantity of sulphurous acid as soon as it attains a red heat, but subsequently again becomes dry, and when fully roasted is in the state of powder. The average time that a charge remains in the furnace is twenty hours.

The refining of the crude metal is conducted in a reverberatory furnace with an iron bed 4 metres long by 2·5 metres broad, covered with a layer of fire-clay 0·28 metre thick. Brown coal is used upon an ordinary grate. The working-door is on one side and the tapping-hole at the end of the bed. The gases on their way to the chimney pass under an iron plate, on which the moulds for the refined metal are placed in order to warm them. The charge consists of 450 kilos. of No. I. metal and 50 kilos. No. II., the first being coarsely broken; while the second is in powder, to which are added 42 kilos. of sulphate of soda, 5 kilos. of charcoal-dust, and 150 kilos. of unroasted ore. The operation lasts ten hours, and a final slag is obtained by the addition of 3·40 kilos. of carbonate of potash and 2·60 kilos. of carbonate of soda, 1 kilo. of raw and 6 kilos. of calcined crude regulus (eliquated sulphide of antimony), and 12 kilos. of the same slag from previous operations. The refined antimony (star-metal) contains as impurities:—

| | |
|---|---|
| As | 0·330 |
| Fe | 0·052 |
| Ag | 0·006 |
| S | 0·720 |

The brittleness of this metal prevents its being extensively employed in a pure state, but its alloys are much used. The most important of these is type metal. Antimony, in the form of a soluble tartrate of antimony and potassium, is the tartar emetic of the apothecary. Antimony, with a mixture of lead, forms type metal and the alloy on which music is engraved. A similar mixture is much used in alkali-works in making pumps and taps for raising and drawing off acids.

## ARSENIC.

Arsenic is a brittle metal, of a steel-grey colour, possessing a strong lustre. When heated to 180° C., it sublimes without first entering into fusion, and when oxidation takes place at the same time emits an odour resembling that of garlic. In close vessels it may be sublimed without change; but, if air be admitted, it is rapidly converted into arsenious oxide. When exposed to air and moisture, it usually acquires, on its surface, a dark film, which is only superficial; it has, however, been observed that some specimens may be kept in open vessels for several years without losing their lustre, while others are in a short time oxidized throughout their whole substance, and fall into powder. This difference has been accounted for by supposing arsenic to exist in two allotropic conditions, but it is equally possible that difference of structure alone may sufficiently explain the phenomenon. The arsenic deposited in the hotter parts of the receivers in which it is collected is of a whiter colour than the other, and is highly crystalline; that found in the cooler portions of the receivers is amorphous, darker in colour, and more readily oxidized.

Arsenic is combustible, and burns with a bluish white flame, and the formation of arsenious oxide $As_2O_3$. This oxide, generally known by the name of "white arsenic," is the most common preparation of this metal. It is obtained by roasting, in a reverberatory or other furnace, ores containing arsenic.

Ores of Arsenic.—The white arsenic of commerce is procured during the treatment of arsenical ores of tin, cobalt, and copper, and to some extent also by roasting arsenical pyrites.

ASSAY OF ARSENICAL ORES.—Digest the finely pulverized ore in strong nitric acid until all action, on the addition of fresh acid, is at an end. Evaporate nearly to dryness, to expel excess of nitric acid, dilute moderately with water, and filter. The filtrate will contain the arsenic in the form of arsenic acid, probably with sulphuric acid, resulting from the oxidation of sulphur. Solution of nitrate of lead is added, and a mixture of arsenate and sulphate of lead thrown down. The liquid is removed by decantation, and the precipitate digested with weak nitric acid by which the arsenate of lead is dissolved, while the sulphate of lead remains. This is separated by filtration, and the filtrate neutralized with sodium hydrate, which throws down arsenate of lead; this must be collected, dried, and weighed. Every 100 parts of this precipitate corresponds with 22·2 of metallic arsenic, or 29 parts of arsenious oxide. The above process, which is comparatively expeditious, is recommended by Mitchell in his "Manual of Assaying," but the writer has not had occasion to test the accuracy of the results obtained. Arsenic may be estimated with considerable accuracy as an ammonio-magnesian arsenate; but in the presence of iron and various other bodies, the operation becomes a process of quantitative analysis, for which some text-book on that subject should be consulted.

MANUFACTURE OF WHITE ARSENIC.—None of the makers of white arsenic in Devon or Cornwall, excepting the Great Devon Consols Company, manufacture it directly from raw ore; the grey flue-deposit from the tin mines is generally employed. At Devon Consols raw ore is roasted in the Oxland and Hocking calciner, and much of the arsenic produced is, without further preparation, sufficiently pure for the market. When any of it is found to be a little grey in colour it is re-sublimed.

A common reverberatory furnace is used for the re-sublimation of crude arsenic, but to prevent discolouration by smoke, either coke or anthracite is used as fuel. A large proportion of the sublimed white arsenic is deposited in an amorphous state, but some of it is found in the form of octahedral crystals. The whole is ground between French burr-stones, and packed in casks for the market. All the white arsenic produced in this country comes from Devon and Cornwall.

Freshly prepared arsenious oxide is perfectly colourless and transparent, but becomes opaque by exposure. It is largely employed by glass-makers, and in the manufacture of emerald green. The annual production of arsenious oxide in this country is about 5,000 tons.

PREPARATION OF METALLIC ARSENIC.—Arsenious oxide when heated with carbonaceous matter becomes reduced to the metallic state. Metallic arsenic is prepared either by the reduction of arsenious oxide, or by the direct decomposition of arsenical pyrites at a high temperature in retorts, which are usually arranged in a gallery.

At Altenberg, Saxony, a mixture of 1 cwt. of arsenious oxide with ½ a cubic foot of powdered charcoal, is submitted to reduction and sublimation in iron retorts. A charge so constituted necessitates the consumption of 8 cubic feet of large coal and 3½ cubic feet of slack.

At Reichenstein, in Silesia, the *löllingite* ($Fe_2As_3$) operated on contains about 23 per cent. of arsenic. The furnaces employed for working these ores have each twenty-six glazed earthen tubes or retorts, varying in length from 26 to 28 inches, and 5 inches in diameter; they are provided with receivers resting upon masonry supports. Of these twenty-six retorts, thirteen are placed on either side of the furnace, seven in a lower row and six in the intermediate spaces above. The chamber enclosing the receivers is provided with an iron door, which is kept shut till the close of the operation; the products of combustion escape through apertures in the arch, which is covered by a conical chimney, like that placed over a steel cementation-furnace. Each furnace is charged with 5 cwts. of ore, which in the course of ten hours yield 100 lbs. of arsenic, of which about 90 lbs. are sold in the light-coloured crystalline state, and the remainder as dark-grey arsenic.

Arsenic is used in small quantities in the preparation of various alloys, and particularly in the manufacture of shot. When a small quantity of this metal is mixed with lead, it is found not only to impart to it a certain degree of hardness, which is advantageous, but it likewise gives it a tendency to form into regular globules, which facilitates the manufacture.

## ZINC.

Although ores of zinc have been employed from remote antiquity for the purpose of converting copper into brass, the metal itself does not appear to have been known in Europe prior to the commencement of the sixteenth century; as we find it first distinctly mentioned by Paracelsus, who died in 1541. It is, however, stated by Beckmann and others to have been first described, under the name of *marchasita aurea*, in the thirteenth century by Albert of Bollstädt, commonly known as Albertus Magnus. Its colour is bluish white, and when recently broken it presents a brilliant crystalline surface. At ordinary temperatures zinc is a brittle metal, but when heated to between 100° and 150° C. it becomes both ductile and malleable. When the heat is increased to about 205° it is again brittle, and may at this temperature be readily pulverized in an iron mortar. Zinc fuses at about 412° C., and when slowly cooled exhibits a highly crystalline texture. The zinc of commerce, known as *spelter*, is not chemically pure, but is invariably contaminated with the presence of various other metals, such as lead, cadmium, and iron. The following analyses of commercial zinc from various localities, given by Schützenberger, show that lead is its chief impurity :—

| | Silesia. | Bleiberg. | Pennsylvania. | New Jersey. | Illinois. |
|---|---|---|---|---|---|
| Zn . . . . | 97·471 | 98·054 | 99·982 | 99·976 | 99·378 |
| Pb . . . . | 2·393 | 1·563 | ... | ... | 0·503 |
| Cd . . . . | traces | 0·282 | ... | ... | 0·078 |
| Fe . . . . | 0·136 | 0·101 | 0·018 | 0·024 | 0·041 |

Belgian zinc contains about 0·3 per cent. of lead, and is free from arsenic. The proportion of arsenic contained in commercial zinc does not usually exceed 0·000097 per cent. It is therefore evident that the zinc of commerce is sufficiently pure for ordinary purposes.

When, however, it is to be employed for the purposes of toxicological investigation it is essential that it should be free from both arsenic and sulphur, and some process of purification becomes necessary. This may be accomplished by a method recommended by Gunning, which furnishes zinc containing a little lead, but perfectly free from arsenic and sulphur. In order to purify zinc by this method, granulated zinc and a mixture of sodium carbonate with sulphur are arranged in alternate layers in a crucible. This is heated until its contents have become fused, and the metal, freed from the adhering slag, is again melted with a little litharge, by which the last traces of sulphur are removed.

Instead of operating as above described, a mixture of four parts of zinc filings and one of nitre may be heated together in an earthen crucible. Deflagration takes place, all the arsenic and a portion of the zinc becomes oxidized, and the unoxidized zinc collects in the form of a button at the bottom of the crucible. Distillation in an earthen retort removes any remaining traces of iron.

Zinc thus treated still, however, retains traces of lead, and to obtain the metal in a perfectly pure state, it is necessary to distil a mixture of pure zinc oxide and sugar-charcoal in a porcelain retort.

When a brilliant surface of clean and polished zinc is exposed to dry air, it remains unchanged at common temperatures; in a damp atmosphere, on the contrary, it is tarnished and soon acquires a grey colour from the formation of a superficial coating of oxide. When heated in contact with air at a temperature sensibly above its point of fusion, it takes fire, and burns with an extremely vivid white flame. The brilliancy of this flame is caused by the combustion of metallic zinc, which gives rise to the formation of zinc oxide, ZnO, the *flores zinci*, *nil album*, or *lana philosophica* of the early chemists, a body which may, practically, be considered as fixed at all temperatures. This becoming white hot communicates to the flame its peculiar intensity of colour. Oxide of zinc obtained by this means is largely employed, when ground with oil, as a pigment, in lieu of white lead; from its perfect whiteness, as well as from the circumstance of its not becoming blackened by sulphuretted hydrogen, it is for many purposes to be preferred to the different preparations of lead.

Zinc is soluble in hydrochloric and in dilute sulphuric acid, with evolution of hydrogen gas; the action of these acids is more energetic on ordinary commercial zinc than on that which is chemically pure. This metal decomposes water with the formation of zinc oxide and the evolution of hydrogen; when zinc is in a state of fine division this reaction commences at a temperature slightly exceeding 100° C. The very finely divided metal which is condensed in the sheet-iron drums fixed upon the clay adapters of zinc-retorts, known by the name of *gris de zinc*, possesses remarkable properties, due to its extreme state of division, and is employed as a reducing agent in various chemical operations. This material, although containing various impurities, decomposes water even at ordinary temperatures. A specimen of this substance, analysed by Aldendorf, afforded the following results:—

| | |
|---|---|
| Zn | 39·90 |
| Pb | 2·47 |
| Cd | 4·09 |
| ZnO | 49·76 |
| $ZnCO_3$ | 3·29 |
| Insoluble residue | 0·39 |
| | 99·90 |

The above analysis may be regarded as fairly typical of this product, since the samples usually differ only with respect to the relative amounts of zinc and zinc oxide present.

Zinc is also soluble, with liberation of hydrogen, in boiling solutions of potash or soda. If, at the same time that the zinc is inserted in the alkaline solution, a slip of iron be placed in contact with it in the same liquid, the decomposition of water may be effected at ordinary temperatures. In this case the zinc alone is attacked, the iron merely serving as the negative element of a voltaic couple, by the action of which the decomposition is much facilitated.

Zinc is largely employed in the form of sheets for covering houses and for making gutters and spouts; also for casting imitation bronzes, for galvanizing sheet-iron, and as a constituent of brass.

## ZINC ORES.

Zinc usually occurs in combination with either sulphur, oxygen, carbonic acid, silica, or sulphuric acid, and is also occasionally found associated with alumina, as in a variety of the species spinel. Before the blowpipe the ores of zinc are almost completely infusible, but when strongly heated on a charcoal support, give off, with greater or less facility, fumes of zinc, which becoming oxidized are deep yellow when hot, and are deposited on the cooler parts of the charcoal as a white incrustation.

The zinc of commerce is chiefly obtained from the natural carbonates and silicates of this metal, and from the native sulphide, or blende. The ores of zinc occur either in veins traversing the older rocks, or in floors and bunches in more recent formations. The first mode of occurrence is perhaps the most frequent, but the more recent deposits are generally most productive.

Native Zinc.—A specimen of native zinc is stated to have been found near Melbourne, Victoria, in a cavity in basalt; but the occurrence of this metal in the native state requires confirmation.

Red Zinc Ore; *Zincite; Zinc oxydé ferrifère*; *Zinkoxyd.* Hexagonal.—Red oxide of zinc, ZnO, although rarely occurring in the crystalline form, has sometimes been met with in crystals belonging to the hexagonal system. It is found at Mine Hill, Stirling Hill, &c., in New Jersey, where it is associated with franklinite, willemite, and calcite.

An analysis of this mineral by Whitney gave:—

| | |
|---|---|
| $ZnO$ | 96·19 |
| $Mn_2O_3$ | 3·70 |
| | 99·89 |

Its specific gravity varies from 5·4 to 5·7; lustre adamantine; affords when scratched an orange-yellow streak; colour red, of various hues, sometimes inclining to yellow. It possesses distinct cleavages, is brittle, and presents a subconchoidal fracture.

Alone, before the blowpipe, this mineral is infusible, but with the addition of borax a yellow transparent glass is obtained. Its surface becomes dull, and ultimately white, by exposure.

A specimen of this mineral, of extreme purity, and weighing 16,400 lbs., was forwarded to the Great Exhibition of 1851.

Zinc Blende; *Zinc sulfuré*; *Blende.* Cubic, tetrahedral.—This mineral occurs either massive, or in dodecahedra, octahedra, and allied forms. Streak white to reddish-brown. Colour, resin-yellow to dark-brown or black; specimens having a green or a red tint are occasionally met with. Lustre waxy or resinous, and when recently broken a brilliant and frequently submetallic surface is obtained. Specific gravity 3·7 to 4·2. This ore, when of a dark colour, contains sulphide of iron, and the red variety is sometimes associated with as much as 5 per cent. of cadmium sulphide. When heated alone, or with the addition of borax, before the blowpipe, it is infusible; when a charcoal support is employed, it yields metallic fumes, resulting in a deposit of oxide of zinc.

Blende occurs in rocks of all ages, and is often associated with ores of lead, as also, though less frequently, with those of iron, copper, tin, and silver. The blende found in this country is, from the amount of iron sulphide it contains, usually of a dark colour, and is hence called *Black Jack* by English miners. This sulphide is found abundantly in Cornwall, Cumberland, and Derbyshire, as well as in Transylvania, Hungary, the Harz, and elsewhere.

A transparent variety of a bright-yellow colour accompanies bournonite and fahlerz at Kapnik in Transylvania; still more beautiful specimens of an olive-green tint are procured from Schemnitz in Hungary; transparent red blende occurs in Spain; whilst Sweden, Bohemia, and Saxony furnish brilliant brown and black crystals of this mineral.

The zinc and sulphur of which this mineral is composed are combined in the proportion of 1 : 1, and its composition will consequently be expressed by the formula ZnS.

Two analyses of this substance from different localities afforded the following results:—

| | Sulphide of Zinc, in Crystals; Arfvedson. | Lamellar, from England; Berthier. |
|---|---|---|
| Zn . . . . . | 66·34 | 61·5 |
| Fe . . . . . | ... | 4·0 |
| S . . . . . | 33·66 | 33·0 |
| | 100·00 | 98·5 |

From the difficulties formerly experienced in its metallurgical treatment, this mineral was until a comparatively recent date but sparingly employed as an ore of zinc, although after careful roasting, it yields, by distillation with carbonaceous matter, a large proportion of the metal it contains.

CARBONATE OF ZINC; *Calamine; Smithsonite; Zinc carbonaté; Zinkspath.* Rhombohedral.—This substance is found in crystals, in concretionary and compact masses, and in pseudomorphic forms. When pure, its colour is yellowish-white: but when much contaminated with iron, it is frequently brown or reddish-brown.

Lustre vitreous, inclining to pearly; streak white; cleavage parallel to the faces of the rhombohedron. Specific gravity from 4·00 to 4·45. Smithson, who analysed two specimens of this mineral from Derbyshire, found them to contain:—

| | 1. | 2. |
|---|---|---|
| ZnO . . . . . | 65·20 | 64·64 |
| $CO_2$ . . . . . | 34·80 | 35·36 |
| | 100·00 | 100·00 |

The composition of this mineral is represented by the formula $ZnO.CO_2$, or $ZnCO_3$. It is soluble in acids with evolution of carbonic anhydride; when strongly heated before the blowpipe, that gas is eliminated and oxide of zinc remains. This is one of the most important ores of zinc, and, together with the silicates with which it is almost invariably associated, is extensively employed for the production of that metal. A compact, fibrous, semi-transparent variety of this mineral, of a pale-yellow colour, and disposed in concentric laminæ, occurs at Alston Moor in Cumberland, where it is found associated with blende and galena in calcareous rocks. It is likewise found in Derbyshire, as also in the Island of Sardinia, in Greece, in Siberia, Hungary, Silesia, Carinthia, and near Aix-la-Chapelle, as well as in many parts of the United States of America.

SILICATE OF ZINC; *Electric Calamine; Zinc oxydé silicifère.* Rhombic, hemimorphic.—This mineral was for a long time confounded with carbonate of zinc, although they differ materially from one another both in their chemical and physical properties. It occurs in mammillated, botryoidal, and fibrous forms; also massive, granular, and crystallized.

Its usual colour is white, with sometimes a bluish or greenish shade, also yellowish to brown. Streak white. Transparent or opaque; vitreous lustre and uneven fracture. Specific gravity 3·3 to 3·6. Crystals of this mineral, when heated, become electric, and the same effect is sometimes produced by friction.

Specimens, analysed by Hermann and Schnabel, gave the following results:—

| | From Nertschinsk. Hermann. | From Santander. Schnabel. |
|---|---|---|
| $SiO_2$ . . . . | 25·96 | 23·74 |
| $ZnO$ . . . . | 65·66 | 66·25 |
| $H_2O$ . . . . | 8·38 | 8·34 |
| $Fe_2O_3$ . . . . | ... | 1·08 |
| | 100·00 | 99·41 |

The composition of this mineral may be represented by the formula $Zn_2SiO_4.H_2O$, or $2ZnO.SiO_2.Aq$. This is a valuable ore, and is commonly associated with the carbonate in deposits containing ores of iron and lead, together with sulphide of zinc. Considerable quantities occur at Bleiberg and Raibl in Carinthia, as also at Freiburg in Breisgau, at Rézbánya in Hungary, Tarnowitz in Silesia, and in the neighbourhood of Aix-la-Chapelle. Concentric botryoidal masses are also found in the Mendip Hills, and at Wanlockhead in Dumfriesshire. Pseudomorphous crystals of the same substance occur in some parts of Derbyshire, and at Schemnitz in Hungary. Before the blowpipe it decrepitates, intumesces, and loses its transparency. When reduced to fine powder, it is soluble in hydrochloric and sulphuric acids on the application of a gentle heat, and, on cooling, silica is deposited in a gelatinous state.

WILLEMITE; *Anhydrous Silicate of Zinc; Troostite.* Rhombohedral. —This mineral occurs in minute crystals, massive, disseminated in grains, and fibrous. Lustre, vitreo-resinous. Colour, whitish- or greenish-yellow, to green or brown, when impure; streak uncoloured; transparent to opaque; specific gravity, 3·89—4·27; brittle, with subconchoidal fracture.

The following analyses of specimens, from Stirling, N.J., are by Vanuxem and Keating, who, together, first described this mineral:—

| | 1. | 2. |
|---|---|---|
| $SiO_2$ | 25·44 | 25·00 |
| $Mn_2O_3$ | 6·50 | 2·66 |
| $Fe_2O_3$ | | ·67 |
| ZnO | 68·06 | 71·33 |
| | 100·00 | 99·66 |

From analyses of this mineral the formula $2ZnO.SiO_2$, or $Zn_2SiO_4$, may be deduced. Willemite occurs at Vieille-Montagne, near Moresnet, and at Stolberg, near Aix-la-Chapelle; at Raibl, in Carinthia; in Servia, and in Greenland. In New Jersey, at Franklin and Stirling, it occurs in such quantities as to constitute an important ore of zinc. A variety known as *Troostite* is found in the last-named localities, in crystals of as much as 6 inches in length by 1 inch in diameter, imbedded in franklinite or calcite. The following minerals, although interesting to mineralogists, are not found in sufficient quantities to be of any metallurgical importance:—

SULPHATE OF ZINC; *Goslarite; White vitriol; Zinc sulfaté; Goslarit.* Rhombic.—Its formula is $ZnO.SO_3.7H_2O$, or $ZnSO_4.7H_2O$. It is a soluble salt of a white colour, and is usually associated with blende, by the oxidation of which it is probably produced. It occurs at Holywell in Wales, at Goslar in the Harz, at Fahlun in Sweden, and at Schemnitz in Hungary.

OXYSULPHIDE OF ZINC; *Voltzite; Leberblende.*—In implanted spherical globules; colour, dirty rose-red to yellow. Its composition may be expressed by the formula 4ZnS.ZnO. A rare mineral occurring at Pontgibaud in France, Joachimsthal in Bohemia, and in Cornwall.

A hydrous phosphate and an anhydrous sulphate of zinc are also stated to occur.

## DISTRIBUTION OF ZINC ORES.

Of all the minerals containing zinc four only are of much general importance to the metallurgist; these are blende, calamine, and the silicates of zinc. These ores are derived from two distinct sources, namely, from mineral veins and from irregular deposits. First, those which occur in the older formations in veins, generally associated with other metallic sulphides; blende is the most abundant ore of this class. The second group, which includes the most important ores, occurs chiefly in calcareous or dolomitic rocks, often belonging to the Carboniferous system. These ores are usually either intercalated with the strata, or occur in the form of irregular masses. By far the largest proportion of the zinc ore raised in Europe is obtained from Greece, Germany, Belgium, Spain, Sweden, and the Island of Sardinia; that produced by other countries being comparatively unimportant. The chief deposits of Germany are situated in Upper Silesia, some also occurring in the Rhenish provinces and in Westphalia. The most important Belgian deposits are those belonging to La Vieille-Montagne, La Nouvelle-Montagne, and the Société Métallurgique Austro-Belge. The Mineral Statistics for 1884 show that the quantity of zinc ores raised in the United Kingdom during that year amounted to 25,563 tons, equivalent to 9,919 tons of metallic zinc; of this quantity 5,364 tons were raised in England, chiefly in the counties of Cornwall, Devon, Cumberland, and Salop; 14,467 tons were obtained from Wales, chiefly from Denbighshire, Cardiganshire, Flintshire, and Montgomeryshire. The Isle of Man during the same year produced 5,685 tons; Ireland, 47, and Scotland none. The principal zinc-works in this country are situated at Swansea, and in the counties of Durham, Denbigh, and Flint. The ores treated include blende, chiefly obtained from various lead mines, together with calamine and silicate of zinc. The two latter are, for the most part, the produce of foreign mines, considerable quantities of carbonate being imported from Spain, Greece, and Sardinia, while silicates are, to a small extent, derived from other sources. The United States of America annually yield large quantities of zinc ore.

The following table gives the production of spelter (crude zinc) for the several years indicated. The data relating to the production of zinc in Europe are those of M. Saint-Paul de Sinçay, Director-general of the Vieille-Montagne Co.; the American statistics are derived from official returns.

PRODUCTION OF SPELTER.

| | 1860. | 1865. | 1870. | 1875. | 1880. | 1882. |
|---|---|---|---|---|---|---|
| | Tons. | Tons. | Tons. | Tons. | Tons. | Tons. |
| Upper Silesia . . | 40,354 | 35,430 | 36,518 | 43,123 | 65,437 | 69,846 |
| Rhenish Province and Westphalia . . | 8,592 | 16,647 | 18,006 | 25,396 | 27,107 | 35,546 |
| Vieille-Montagne . | 28,925 | 30,592 | 42,112 | 41,618 | 44,690 | 48,861 |
| Other Belgian Works. | 9,144 | 13,485 | 14,476 | 18,836 | 26,700 | 35,625 |
| Société Asturienne Spain . | 1,777 | 1,325 | 3,048 | 3,000 | 4,000 | 5,047 |
| France . | ... | ... | ... | 5,311 | 8,591 | 11,423 |
| Other French Works . | ... | 500 | 500 | 1,500 | 3,000 | ... |
| England . . . | 6,104 | 6,523 | 16,000 | 15,903 | 22,000 | 25,581 |
| Poland . . . | 1,500 | 3,000 | 3,625 | 3,000 | 4,463 | 4,544 |
| Austria . . . | 1,500 | 1,000 | 1,000 | 1,000 | 3,199 | 3,199 |
| Totals for Europe . | 97,896 | 108,502 | 135,285 | 158,687 | 209,187 | 239,672 |
| United States of America . . . . . . | | | | 14,345 | 21,054 | 30,591 |
| Totals for the World . . . . . . | | | | 173,032 | 230,241 | 270,263 |

## ASSAY OF ZINC ORES.

The zinc-yielding materials which are likely to come under the notice of the metallurgist may be classified as follows: —

1. Ores of zinc in which the metal exists in the form of oxide, uncombined with silica.

2. Ores in which zinc is present as oxide, but wholly or in part combined with silica.

3. Ores in which this metal exists wholly, or in part, combined with sulphur.

The estimation of the amount of zinc contained in an ore of that metal may be effected either by dry or by humid assay. The determination by the former method is a troublesome and uncertain process, since zinc, being volatile and oxidizable at high temperatures, cannot be obtained in the form of a metallic button, as in the case of the more fixed metals.

FIRE ASSAY.—Ores of the first class, being readily reducible, only require to be mixed with carbonaceous matter and heated to whiteness in a porcelain retort. The metal is by this means volatilized and condenses in the neck of the retort, which, during the operation, must be kept open by the occasional insertion of an iron rod, since otherwise an explosion might ensue. When the operation is completed, which is usually the case in about an hour, the retort is removed from the fire and laid on its side to cool. It must then be broken, and the zinc detached from the neck as completely as possible, together with any globules which may have condensed in the dome; this separation is more readily effected if the neck has been previously coated with plumbago. Any zinc which cannot be thus mechanically detached, must be removed by immersing the fragments of the retort in hot nitric acid; the solution thus obtained is evaporated to dryness, the residue, of nitrate of zinc, being converted into oxide by ignition. This is weighed, and four-fifths of its weight added to that already obtained by weighing the zinc obtained in the metallic state.

Ores of the second class are likewise reducible by carbon, but it is found desirable to add some base, such as lime, which combines with the silica set free on the reduction of the zinc oxide, forming with it a fusible slag.

Ores of the third class must, previously to treatment by the same process as those of class 1, be subjected to careful roasting to free them completely from sulphur. To prevent clotting, the temperature must be kept low during the first portion of the process, but must be considerably raised towards the end, in order to insure decomposition of any sulphates which may have been formed.

HUMID ASSAY. BY DIFFERENCE.—This process is based on the solubility of zinc oxide in solutions of ammonia and ammonium carbonate, and is not suitable for ores in which the zinc is present as silicate, upon which the above-named reagents have but little effect. On this account the process may be found useful in determining, approximately, the amount of zinc carbonate present in an ore in which that metal also exists as silicate. The process is conducted as follows:—A weighed quantity of the ore is calcined in a muffle or over a gas-flame until all volatile matters have been expelled. The calcined ore is then re-weighed, placed in a beaker or flask, and digested for about half an hour with a solution of ammonia and ammonium carbonate.

The insoluble residue is collected on a filter, thoroughly washed with hot ammoniacal water, dried, ignited, and weighed. This weight deducted from that of the calcined ore employed gives, by loss, the weight of zinc oxide; from which, by subtracting one-fifth for oxygen, we get the weight of zinc present as carbonate in the ore operated upon.

If the ore be in a state of fine division and tolerably pure, the zinc oxide may be almost completely extracted by ammonia and ammonium carbonate. If, however, much earthy matter or oxide of iron, more especially the latter, be present, some of the zinc oxide is retained in the insoluble residue. To obviate this the process may be advantageously modified as follows:—After calcination, the ore is treated with hydrochloric acid and a little nitric acid; to the solution obtained ammonia and ammonium carbonate are added in excess. The insoluble residue, together with the precipitate of ferric hydrate, &c., is collected on a filter, washed with hot ammoniacal water, dried, ignited, and weighed. The difference between this weight and that of the calcined ore used indicates, approximately, the quantity of zinc oxide dissolved. By this treatment, not only the zinc originally existing as carbonate, but also that present as silicate, is removed; silicate of zinc being decomposed by the acids employed.

VOLUMETRIC ASSAY.—The volumetric estimation of zinc by a standard solution of sodium sulphide is one of the quickest and most reliable methods known, and is therefore commonly used for commercial purposes. When sodium sulphide in solution is added to an ammoniacal solution of zinc, the latter metal is thrown down as a white precipitate. By using a standard solution of sodium sulphide, the quantity of zinc present may be calculated from the amount of the solution required for its precipitation. Various methods have been employed to determine when the whole of the zinc has been precipitated, the following being those most usually resorted to. The first of these will generally be found very convenient.

Flakes of freshly precipitated ferric hydrate may be produced in the ammoniacal solution of zinc by the addition of ferric chloride; these will retain their colour until all the zinc has been precipitated, but will blacken directly the sodium sulphide is in excess. Or a drop of the zinc solution may be transferred, by means of a glass rod, to a piece of lead-paper, when the presence of excess of sodium sulphide will be indicated by the blackening of the paper. Instead of this a drop of a solution of pure nickel sulphate, or lead acetate, may be placed on a white porcelain slab and touched with a glass rod, which has previously been dipped into the zinc solution. If there be any excess of sodium sulphide, a black precipitate will be formed; if, on the contrary, a portion of the zinc still remains in solution, no change will occur.

The preparation of the necessary re-agents is conducted as follows:—

*Ferric Chloride Solution.*—3 grammes of iron-wire are dissolved in hydrochloric and a little nitric acid, and the solution diluted to one litre. 5 c.c. of this solution will contain about 0·015 gramme of metallic iron.

*Solution of Sodium Sulphide.*—About 100 grammes of crystallized sodium monosulphide are dissolved in about 2½ litres of distilled water. The solution should be either filtered or decanted, in order to free it from any black precipitate which, in small quantities, may be formed; it is then ready for use. Standardization is effected by dissolving two or three pieces of zinc, each weighing about 0·5 gramme, in hydrochloric acid containing a small quantity of nitric acid; each solution is put into a separate flask and diluted with water to about half a litre; ammonia in excess is then added. Into each flask 5 c.c. of ferric chloride solution, to which ammonia has been previously added, are introduced. Solution of sodium sulphide is now allowed to run slowly from a graduated burette into the cold ammoniacal solution, which must be kept in constant motion, until the zinc is completely precipitated and the ferric hydrate is blackened. The number of divisions of the liquid which have been run into each flask is then read off, and the mean of the results taken as the standard. From the quantity of this solution required to precipitate a known weight of zinc, the weight of that metal which each c.c. of sulphide solution will precipitate is readily found.

When sodium monosulphide is unobtainable about 300 grammes of caustic soda may be dissolved in 1 litre of water, and the solution divided into two equal portions. Through one of these, sulphuretted hydrogen is passed to saturation; the other portion of the solution is then added, and a preliminary trial of the strength of the mixture is made. It may then be diluted to approximately the required strength, and standardized in the usual way.

*Estimation of the Zinc contained in an Ore.*—From 0·5 to 2 grammes of the pulverized ore are heated with hydrochloric and a little nitric acid, a large excess of acid being avoided. When the decomposition is complete, the solution is somewhat diluted, and ammonia and ammonium carbonate added in excess; it is then kept at a gentle heat for twenty or thirty minutes, filtered into a flask, and the residue on the filter well washed with hot ammoniacal water. To the ammoniacal solution 5 c.c. of the ferric chloride solution should be added. It is, however, preferable first to add ammonia to the 5 c.c. of ferric chloride solution, and then to pour the liquid containing precipitated ferric hydrate into the ammoniacal zinc solution; this mode of procedure prevents the coagulation which would be liable to occur if the ferric chloride were directly added. When the solution is cold the sodium sulphide solution is gradually run in from a burette, until the zinc is completely precipitated and the ferric oxide is slightly blackened. The number of divisions of the solution run in is now read off, and the weight of zinc which that quantity of the standard solution is capable of precipitating is determined by calculation; the result being the quantity of zinc contained in the weight of ore employed. During the addition of the sodium sulphide solution the contents of the flask must be kept in continual agitation, otherwise the zinc may be entirely precipitated, in one portion of the solution, and the ferric hydrate blackened, before the precipitation of the whole of the zinc has been effected. Should this occur, the results obtained would be entirely unreliable.

Ores of zinc are frequently contaminated by the presence of other metals, which would vitiate the assay unless previously removed.

*Iron.*—When this metal is present in small quantities only, the assay may be made without filtration; it is, however, preferable to filter off the ferric hydrate from the solution of zinc in ammonia and ammonium carbonate, and to add to the filtrate a definite amount of the ferric chloride solution. When the quantity of iron is considerable the ferric hydrate is liable to retain zinc oxide; in this case, after allowing the ferric hydrate to subside, the ammoniacal solution is decanted, and the former again treated with ammonia and ammonium carbonate before filtration.

*Manganese* is sometimes present in red zinc oxide, &c. On the addition of ammonium carbonate, the manganese is not completely separated; but its complete precipitation may be effected by the application of heat and the addition of a few drops of bromine to the ammoniacal solution.

*Copper*, if present in small quantities, communicates a blue colour to the solution; it may be removed by the addition of a few drops of sodium sulphide solution to the ammoniacal zinc solution, whilst hot, until the blue colour is discharged. The precipitated copper must be separated by rapid filtration, and the zinc estimated in the filtrate, after cooling, in the usual manner. If copper be present in large quantity, the ore must be treated with dilute sulphuric acid, and the copper thrown down by immersing in the solution a clean plate of iron. The precipitated copper is thrown on to a filter, and, if necessary, may be washed, dried, and weighed. To the solution a small quantity of nitric acid is added, to peroxidize the iron, and after adding ammonia and ammonium carbonate the assay is proceeded with in the usual manner.

*Lead and Cadmium* do not interfere with the accuracy of the results, since the former remains partly in the insoluble residue, as sulphate, and the remainder is precipitated on the addition of ammonium carbonate, whilst cadmium is insoluble in the ammoniacal solution.

*Silver* is rarely present in sufficient quantity to interfere with the assay, but if necessary it may be separated by filtering the dilute solution of zinc oxide in hydrochloric acid, previous to adding ammonia; the silver will thus be left with the insoluble residue on the filter, in the form of chloride.

## METALLURGY OF ZINC.

When calamine is the ore operated on, it is first submitted to calcination, for the purpose of rendering it less compact and more readily acted on by the carbon used for its reduction, while at the same time carbonic anhydride and water are driven off. In some cases the larger fragments of ore are roasted in kilns, the calamine being interstratified with layers of fuel. A modified form of lime kiln having an independent fire-place is used in northern Spain for this purpose. An apparatus of this description will yield from 5 to 8 tons of calcined calamine in twenty-four hours, while one having two grates will calcine as much as 10 tons, with a consumption of from 7 to 9 per cent. of coal. This, though possessing fewer disadvantages than ordinary kilns, is, nevertheless, liable to yield a product of which the degree of calcination is not uniform.

In other cases the calcination is conducted in reverberatory furnaces, but the ore must not be in large fragments, which are both difficult to turn and to thoroughly calcine. The furnaces employed are of the ordinary reverberatory description, and may be heated either by an independent fire-place or by the waste gases from the reduction-furnaces. When the latter are employed, considerable economy of fuel is the result, but the draught and consequently the temperature of the retorts nearest the calciner are liable to be decreased.

Silicates of zinc are not always calcined previous to treatment for the metal they contain, but are frequently mixed with slaked lime, in addition to the coal-dust used for their reduction.

The process of roasting blende, which has for its object the conversion of zinc sulphide into zinc oxide with the elimination of sulphurous anhydride, is by no means so easily accomplished as the calcination of calamine. When blende is roasted zinc sulphate is formed, and this can only be decomposed by a strong heat. Kilns are not suitable for the roasting of blende, but they are occasionally employed for a preliminary heating having for its object the rendering of the blende more friable. Roasting can only be satisfactorily accomplished after the blende has been reduced to a uniform coarse powder.

The calcined or roasted ore is reduced to powder in suitable mills, a coarser powder being used for the Silesian than for the Belgian process. In many cases roasted blende requires no further crushing, having been ground sufficiently fine previous to roasting. The calcined ore is mixed with a proper proportion of finely divided non-caking coal, by which, when strongly heated, the production of metallic zinc is determined. The reduction of the oxide is effected at the expense of the carbon present, carbonic oxide is evolved, and the metallic zinc liberated is condensed in proper receivers adapted to the retorts in which the operation is conducted.

## English Process.

In 1721 Henckel published his discovery of the fact that zinc could be obtained from calamine, and in 1742 Van Swab extracted it from its ores at Westerwick in Dalecarlia, where it was proposed to erect works for its production on a large scale. Dr. Isaac Lawson, in this country, is reported to have first invented a practical method of extracting zinc from calamine, and to have erected works for that purpose. According to Pryce, zinc-works were first erected at Bristol by Mr. John Champion, to whom a patent appears to have been granted in 1739.

The process formerly employed in this country for the reduction of zinc ores is known as distillation *per descensum*, and was conducted in furnaces in many respects similar to those used in glass-houses for the preparation and fusion of glass. These furnaces were either square or round in horizontal section, but that represented in fig. 147, which was usually preferred, has the latter form. The fire-place, F, was raised to a convenient height above the ground, and was situated in the centre of the arrangement. Around this were disposed the pots, *c*, which were charged with the mixture of ore and fine coal from which the zinc was distilled. The conical hood was pierced with openings, *d*, corresponding to each pot, through which the charge was introduced. The bottom of each pot was furnished with an orifice, which was stopped by a plug of wood; this, being converted into charcoal during the process, was rendered sufficiently porous to admit of the passage of the volatilized metal, but at the same time prevented the escape of the small coal or calcined mineral. Each pot was covered by a lid secured in its place by a luting of fire-clay; the volatilized zinc was condensed in a sheet-iron

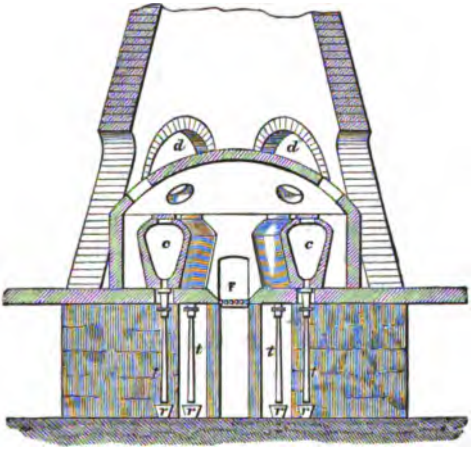

Fig. 147.—English Zinc Furnace; vertical section.

tube fitted beneath the opening in the bottom of each pot. These tubes were made in two portions, the shorter being funnel-shaped and so arranged that it could be raised or lowered in order to keep the flange at its upper end in close contact with the bottom of the pot. The longer piece of tube was fitted loosely on to the shorter, but was not placed in position until the zinc began to distil off. The condensed metal fell, in the form of drops, into an iron vessel, *r*, placed for its reception. As the tubes were liable to become choked by the condensed metal, it was necessary to clear them from time to time by the insertion of a long iron rod, since they might otherwise become entirely closed, and give rise to explosions. The zinc collected in the form of drops and fine powder, mixed with a certain proportion of oxide, was afterwards melted in an iron pot, set in brickwork, and heated by a fire beneath. The dross which collected on the surface of the fused metal was skimmed off and returned to the pots in a succeeding operation, while the zinc itself was cast into rectangular slabs or ingots, in which state it was sent into the market. Five distillations were made by a furnace of this kind in fourteen days, in the course of which 5 tons of calcined ore were treated, yielding about 40 cwts. of metal. About 25 tons of a mixture of binding and free-burning coal were employed in the extraction of 1 ton of metallic zinc. The duration of each pot was calculated at about four months; when unfit for further service, they were removed through apertures made in the surrounding brickwork. New pots, before being set, were heated to redness in a reverberatory furnace, and carried to their places by a large pair of iron tongs slung in chains, and supported by a traveller similar to that used for replacing glass-house pots. When set in their places, the brickwork was repaired and a cover fitted in the usual way. At the close of each operation, the crucibles were discharged by removing the condensing-pipe from the bottom, and withdrawing the residue through the aperture, after breaking with a rake the charcoal by which it was partially closed during the process of distillation. Any cracks which might occur in the pots during the process were stopped with fire-clay.

The consumption of fuel in this apparatus was too large in proportion to the amount of metal obtained to enable it to compete with the Belgian and Silesian furnaces. It is believed that the English zinc-furnace is no longer anywhere in use, although it was seen in operation by Dr. Percy so recently as 1859.

## BELGIAN PROCESS.

Calamine, which also occurs abundantly in certain Devonian and Carboniferous rocks of Belgium, was for a long period obtained from deposits at Moresnet in the neutral territory, and was employed at Stolberg, Liége, Namur, and elsewhere for the manufacture of brass by cementation, but the art of isolating, on a commercial scale, the peculiar metal which by uniting with copper transformed it into brass remained unknown.

In the year 1805, however, the Abbé Dony, who does not appear to have been acquainted with what had been done by others in the same direction, accidentally produced this metal. When endeavouring to extract zinc from calamine, by fusion in a reverberatory furnace, it occurred to him that the heat applied was probably not sufficiently great, and he consequently added coal-dust to the mineral. He also built into the side of his furnace an ordinary flower-pot to serve as a peep-hole through which he might watch the various changes taking place in the interior. On the inside of this flower-pot, which was considerably cooler than the furnace itself, drops of condensed zinc began plentifully to appear, and the method of extracting this metal by distillation from a mixture of calamine and coal-dust, by the Belgian process, was thus established.

In the year 1810 the Abbé Dony associated himself with a M. Chaulet, and erected small works at Liége, but in 1818 they found themselves completely ruined, and were obliged to dispose of their establishment to Dominique Mosselman, whose representatives in 1837 founded the Vieille-Montagne Company.

The first furnaces erected at Liége contained each 16 retorts or *pots*, which number was afterwards increased by Mosselman to 20. The charge of such a furnace was 500 kilos., and the production of metal 200 kilos. per diem. In 1840 it was considered a great improvement had been made by arranging these furnaces in blocks of four, each containing 48 pots, and obtaining a daily production of from 250 to 300 kilos. At the present time, with about the same consumption of fuel, and only a very slight increase in labour-cost, the large furnaces at Angleur, which are the direct descendants of the Dony type, yield from 1,300 to 1,400 kilos. per diem from each face containing 160 pots.

*Calcination of Ores.*—Up to the year 1857, the ores treated at the various Belgian zinc-works consisted exclusively of calamine, with a certain amount of silicate of zinc, but since that period large quantities of blende have been employed. The ores at first used were principally massive, and were calcined in kilns similar to those constructed for burning lime, but when poorer material was enriched by crushing and washing it became necessary to employ reverberatory furnaces, as kilns are not suitable for the calcination of finely divided stuff. Calamine, when heated, loses its water and carbonic anhydride, and is converted into zinc oxide; hydrated silicate of zinc loses its water only. Blende is transformed by roasting into oxide of zinc, while its sulphur passes off in the form of sulphurous anhydride. Calamine may be calcined in kilns with an expenditure of about 7 per cent. of fuel, but in the reverberatory furnace about 10 per cent. of the weight of the ore will be required, and the total cost in Belgium amounts to about 4 fr. 50 c. per ton of roasted ore.

Blende is roasted in reverberatory furnaces, which are usually provided with two beds. The ore to be calcined is first finely ground, and then spread equally over the surface of the upper bed, where it remains twenty-four hours, and is turned over from time to time. After the withdrawal of the former charge the ore on the upper bed is transferred to the lower one, where it is frequently turned over so as to expose every portion of it to the air. The formation of zinc sulphate is to be as much as possible avoided, and for that reason the ore should not come in contact with an unlimited supply of air. When an ore contains pyrites it becomes more or less fusible, and great care is necessary to avoid agglomeration. In order to decompose any sulphate of zinc formed during the operation it is necessary that the charge should be very strongly heated before drawing, and for this purpose it is collected in the portion of the furnace lying immediately before the bridge. This operation is somewhat long and difficult, but as the presence of only 1 or 2 per cent. of sulphur in the ore causes a serious loss during the process of reduction, it is important that it should be removed as completely as possible. The consumption of fuel is about one-fourth the weight of the ore treated, and the total cost is estimated at 15 f. per ton of roasted blende.

Messrs. Hasenclever and Helbig have proposed the use of a furnace by which a portion of the sulphur in the blende could be rendered available for the manufacture of sulphuric acid, but it does not appear to have been generally adopted. When silver is present in blende it is to a considerable extent carried off during the operation of roasting. It is, however, stated that the addition of lime or sodium carbonate to the raw ore to a great extent prevents this loss.

*Pots or Retorts.*—It is as important that zinc-works should be so situated as to command a good supply of refractory material for the manufacture of pots and fire-bricks, as that they should be in the vicinity of cheap coal. The clay employed must be very refractory, and, as a rule, should contain but little free silica; that from Andenne is usually preferred.

The pots for the distillation of zinc were formerly all hand-made, but are now manufactured by machinery, by which they are not only produced at a cheaper rate, but also of superior quality. A mixture of unburnt and burnt clay is, after drying, ground together in a Vapart mill, mixed and kneaded in a horizontal pug-tub, and finally delivered, by an endless screw, upon the moulding-table. The section of the pots is either round or elliptical, and they are made by forcing, by a heavy rammer worked by machinery, the prepared clay into a sheet-iron mould of the proper form, opening longitudinally, and lined with wet canvas. In this way a sausage-shaped mass of clay is obtained 1·30 m. in length, which, while still in the mould, is bored by machinery to within 0·03 m. of its lower end; if round, the interior diameter is at Angleur 0·16 m., or if oval, 0·16 m. × 0·20 m.; the thickness adopted at the same works being 0·03 m. The mould is now opened, the canvas around the outside of the pot removed, and the pot gradually dried so as to be ready for delivery to the zinc-furnace. This is done by successively passing it through a series of stoves of which the temperature progressively increases. Elliptical pots are made in moulds having the form of their exterior, while the interior is cut out by two parallel borings intersecting one another throughout their whole length.

In many of the zinc-works the pots are now made by hydraulic power, and are much more compact and durable than those prepared in any other way. The apparatus employed for this purpose is a strong cast-iron cylinder, fixed vertically, of which the interior diameter is that of the exterior of the pot to be made. In this is an annular ram exactly fitting its interior surface, while within it is a cylindrical ram having the dimensions of the inside of the pot. Both these rams work from the bottom upwards, and the top of the external cylinder is closed by a cover securely keyed in its place. In order to make a pot with this machine, the cover is removed, the two rams lowered together to a proper distance, and a roll of clay, previously prepared, introduced. The cover is now keyed in its place, and the two pistons raised so as to strongly compress the clay. By a proper arrangement of the taps, the solid ram is now made to advance by increased pressure, displacing the clay, which, having no escape, forces back the annular piston and occupies its place. The top is then removed and the annular piston raised, pressing the pot, which can be cut off by a piece of wire at any required length, out of the mould. These pots are usually glazed on the inside by moistening them with a strong solution of common salt before firing.

As an example of the advantages derived from machine-made pots, in addition to their cheapness, it may be stated that the consumption of hand-made pots was formerly from 5 to 7 for every 300 kilos. of zinc produced, whereas at the present time it does not amount to much more than one-twentieth of that proportion.

*Reduction and Distillation.*—Zinc oxide is reduced, at high temperatures, by carbon monoxide, but the reverse takes place at a red heat; metallic zinc reducing carbon dioxide to carbon monoxide. This reaction gives rise to a certain amount of loss, which is reduced to a minimum by the addition of a large excess of small coal.

At a red heat hydrogen reduces zinc oxide with the production of watery vapour, but the reverse reaction may take place at the same temperature. In order that the volatilized zinc should remain in the metallic state it is necessary it should be carried off by a rapid current of hydrogen.

Silicate of zinc is entirely reduced by carbon at a high temperature, but in practice this ore sometimes yields little more than one-half the metal it contains, with the formation of a more acid silicate. To obtain the best results it is necessary that the temperature should be high and the heat long continued.

Zinc being a volatile metal, it is always obtained by distillation The only metallurgical process employed is the reduction of zinc oxide, or silicate of zinc, in close vessels by carbon. A white-red heat is necessary, and this temperature requires to be kept up for a very considerable time. In all cases, however, the reduction is imperfect, the residues sometimes retaining above 10 per cent. of the zinc contained in the ores.

The Belgian process is characterized by the employment of distillatory apparatus, consisting of fire-clay tubes or pots supported at their two extremities by the walls of the furnace in which they are enclosed. The general arrangement of these pots is shown in fig. 148. The pots, *a*, have a length of from 1·20 m. to 1·30 m., with an internal diameter of from 0·16 m. to 0·20 m., their thickness varying from 0·025 m. to 0·040 m. At the mouth of each pot is placed the condenser, consisting of a fire-clay receiver, *b*, with an enlargement for retaining the liquid zinc. To this is attached a sheet-iron drum, *c*, of variable form, which is pierced at its extremity by a small aperture admitting of the escape of the gases generated. In this way the condenser is protected from the air, which would otherwise burn a portion of the zinc produced.

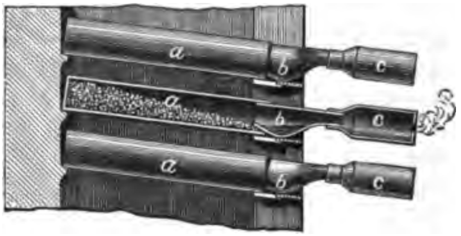

Fig. 148.—Retorts, Receivers, and Drums.

The furnaces are usually built in couples, back to back, the refractory wall between the two being common to both. The fire-chamber is rectangular with vertical sides, and encloses a large number of pots placed across it with their closed ends resting upon ledges in the back wall, while their mouths are supported on fire-tiles resting upon longitudinal cast-iron bearers supported at intervals by vertical pillars.

In this way the front of the furnace is divided into a series of rectangular spaces, each of which encloses one or more pots; these are all slightly inclined from behind forwards. The heating of zinc-furnaces is accomplished in various ways. Formerly they were heated by fire-places situated at a distance of about a metre below the lowest pots, provided with doors, beneath the level of the floor, for the introduction of fuel. The cinders fell into an ash-pit, and were removed through an arched passage constructed for that purpose. The flame passed around and between the pots, and the products of combustion escaped to the chimney through channels situated above the arch of the furnace.

Furnaces are now built much larger than formerly; at Angleur, where they are constructed in couples, they have a fire-place at each end, and contain 160 pots on either side. The pots in the bottom row are not charged with ore, but are left open at the end and perforated with holes to supply heated air for the combustion of unconsumed gases. At this establishment the three lower rows of working pots are oval and placed with their longer axes in a vertical position, while all above them are cylindrical. The first four rows of pots only are provided with wrought-iron drums on their condensers, as the upper ones are not sufficiently heated to require them.

Since the introduction of gaseous fuel for metallurgical purposes, gas-furnaces, in accordance with the systems of Boëtius, Siemens, and others, have been applied to the distillation of zinc with satisfactory results. The Boëtius furnace, of which fig. 149 is a vertical and fig. 150 a transverse section, from an excellent article on zinc by M. F. de Lalande, published in Wurtz's 'Dictionnaire de Chimie,' may contain 147 pots 1·25 m. in length and of the usual diameter, arranged in seven rows of 21 pots each. The furnaces are built back to back in couples.

The cavity of the furnace, which measures 7 metres in length, 2·50 m. in height, and 1·20 m. in depth, is heated by two Boëtius gas producers, A, placed at its two ends. The depth of these is 1·20 m., and their width 0·93 m., the bottom being closed by parallel inclined fire-bars. The fuel is charged through the opening, B, and the producer is constantly kept full of coal up to the charging-door. The supply of air is regulated by horizontal openings, C. The gases generated enter the furnace through the vertical openings, D, which measure horizontally 0·95 × 0·30 m. In the brickwork are built channels, E, which at F are in communication with the arched chamber, G, and at the other extremity are connected with a series of flues constructed in the brickwork below the pots. The cold air from G, passing through these channels, becomes heated, and escaping into the vertical flues at D, meets the gas from the producers.

In order, as much as possible, to equalize the temperature of all parts of the furnace the flame which circulates around the pots, *l*, enters the opening, I, and passing off through the flue, K, the products of combustion escape through the chimney, T. A second portion of ignited gas envelops the pots, *m*, passes through the channels, *n*, beneath the pots, into the flue, *o*, and escapes to the chimney through the arch, P. A third portion of the inflamed gases, rising in the furnace, passes through the channels, *q*, into the flue, R, and thence through S to the chimney, T. It must, however, be remarked that the products of combustion escape at a very high temperature, which results in a considerable loss of heat. This waste heat has sometimes been employed for the calcination of calamine.

At one of the works belonging to the Vieille-Montagne Company, the Boëtius furnaces have been replaced by others heated on the Siemens principle, but without the use of regenerators for heating the air and gas. The large amount of oxide of zinc deposited in the flues through the breaking of pots renders the use of the hollow brickwork regenerator extremely difficult; the gas therefore is taken directly from the producers, while the air is heated by being forced by a fan through channels constructed in the brickwork of the furnace.

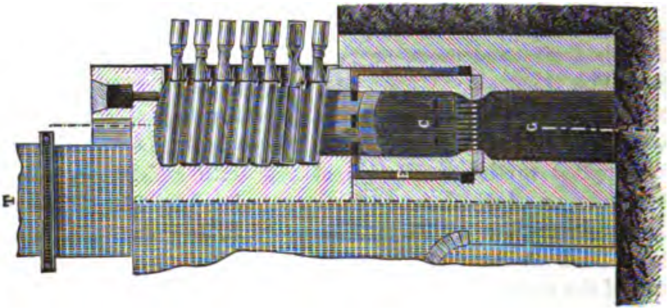

Fig. 150.—Transverse Section.

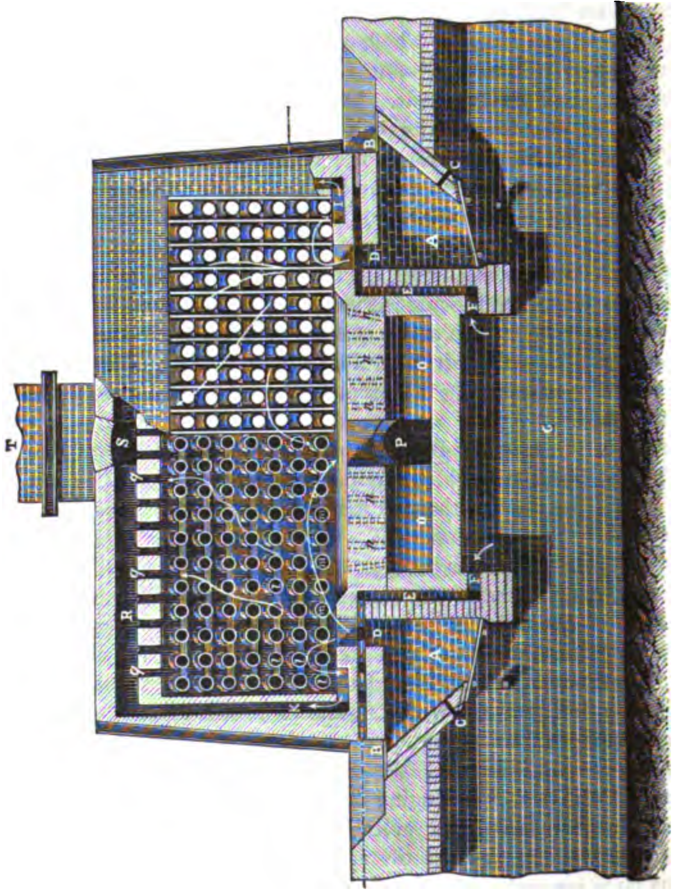

Fig. 149.—The Boëtius Zinc-Furnace; vertical section.

Whatever may be the method of heating employed, the process of reduction remains always the same.

When first fired the apparatus must be heated very slowly, the pots being previously introduced; those required to replace breakages during the progress of the operations are, however, supplied from a special furnace in which they have been previously heated to redness. The calcined ore is finely ground with half its weight of a non-caking coal in a Vapart or other suitable mill, and introduced into the pots by a semi-cylindrical iron scoop. The charge of each pot consists of from 10 to 12 kilos. of ore, and, in order to charge the upper rows, the workmen stand upon a rough wooden table placed before the furnace for that purpose. When the pots have been charged the receivers are luted into their places, and, if necessary, the front of the furnace is repaired with pieces of brick and fire-clay. The sheet-iron drums are not added until after the expiration of two hours.

About this period zinc vapours begin to make their appearance, and to burn with a brilliant greenish flame, at the extremity of the condensers. The first portions of zinc which distil over become partially oxidized, and are deposited as a grey powder in the sheet-iron drums; subsequently the greater portion of the metal condenses in a liquid form in the clay-condensers. At the expiration of about eleven hours the operation is finished, and the workmen proceed to collect the last portions of metallic zinc. For this purpose the sheet-iron drums are removed, and placed with their mouths downwards to prevent the ignition of the finely divided zinc which they contain. Then, with a semicircular iron-rake, the liquid metal is withdrawn from the condensers or receivers into ladles, from which it is transferred to moulds, where it is cast into ingots, each weighing about 20 kilos. The metal is collected either two or three times during the working of each charge. Finally the condensers are removed from the mouths of all the pots, which, after being emptied by the use of an iron scraper, are again filled, and the operation is recommenced. A portion of the pulverulent zinc collected in the iron vessels is ground with oil, and afterwards sold as a pigment. Some of it is used for the reduction of indigo, or for the preparation of hyposulphite of sodium, but the greater part is mixed with coal-dust, and returned to the pots for the purpose of being converted into spelter. Two workmen are required for charging and discharging every 50 pots contained in a furnace, and in case of the mouth of a condenser becoming obstructed it is opened by the introduction of a heated iron rod. The cost of treatment varies slightly at each works, but may be taken approximately at 85 fr. 50 c. per ton of 50 per cent. ore treated. The consumption of fuel varies from six to seven times the weight of the zinc produced, and the loss of metal is from 20 to 25 per cent. of that contained in the ore.

The principal seats of the zinc manufacture in the United States are at Newark, New Jersey; Lasalle, Illinois; Lehigh, Pennsylvania; in Kansas, and at St. Louis and Carondelet, in Missouri. The calcination of calamine is chiefly effected in kilns, and that of blende in reverberatory furnaces. The reduction of the calcined ores is in all cases effected in furnaces of the Belgian type, anthracite being the fuel generally used. This is either burnt directly or in gas-producers, with or without regenerators. The largest furnace in America, containing 408 pots fired by gas without regenerators, is at Matthiessen and Hegeler's Lasalle Works.

The average consumption of coal in producing one ton of zinc at the different American works varies from 5·7 to 7·4 tons, of which from 4·5 to 5·5 tons are burnt, and 1·2 to 1·9 ton added to the charges of ore in the pots.

*Zinc White.*—Zinc white, which of late years has been much employed as a pigment instead of white lead, is prepared either by the combustion of metallic zinc, or directly from ores of that metal. This substance possesses the advantages over white lead of being comparatively innocuous, and of not becoming discoloured when exposed to an atmosphere of sulphuretted hydrogen.

When made directly from zinc the metal is heated in properly constructed vessels to near its point of ebullition, when it takes fire and is gradually converted into a flocculent white oxide, which is carried off by an artificially produced current of air through large sheet-iron tubes into a series of chambers divided by partitions of cotton cloth.. By this treatment the oxide is subjected to a mechanical classification which separates the coarser particles from the finer. Theoretically, 100 parts of zinc should yield 125 parts of zinc oxide.

In the United States of America the manufacture of zinc white is generally carried on by what is known as the Wetherill process. This consists of heating finely crushed zinc ore, which may be mixed with Franklinite and silicates from New Jersey, calamine, or roasted blende, mixed with limestone and small coal, upon the bed of a muffle-shaped furnace. This mixture, which forms a stratum a few inches in depth, is spread upon a layer of coal on a cast-iron plate perforated with numerous holes, through which air at a low pressure is introduced by a fan blower. The zinc thus reduced rises in the state of vapour together with the furnace gases, consisting chiefly of carbon dioxide, but when these have somewhat cooled down, the zinc is re-oxidized partly at the expense of carbon dioxide, which becomes reduced to monoxide. The zinc oxide, in the form of a fine white powder, is carried through a system of chambers in pipes to which coarse cotton-cloth sacks are attached. These act as filters, through which the gaseous products of combustion readily find their way into the atmosphere.

## SILESIAN PROCESS.

In Upper Silesia, where large quantities of zinc are produced, the apparatus employed differs materially from that used for the Belgian process. Its form will be understood by reference to figs. 151, 152, and 153,[1] which represent a furnace erected some years since at Llansamlet, near Swansea.

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[1] From Percy's 'Metallurgy,' by permission of the author.

The bed is flat, horizontal, and very nearly square; on each side of it rise six similar and equal arched recesses, *a*, fig. 153. In front is a vertical wall, *b*, in which is the fire-hole, *c*, fig. 152; at the back there is a similar wall, *d*, behind which rises the stack, *e*; the furnace is covered by an arch, *f*, extending from side to side. Below the fire-place is an arched passage, *g*, which serves for the admission of air to the fire, and allows the stoker to get ready access to the grate. The vertical partitions, *h*, are made of large single fire-tiles. A series of rectangular niches, *i* (fig. 153), correspond with the lateral arched recesses in the upper part of the furnace; these are covered by cast-iron plates, with the exception of a small opening, through which the iron adapters of the retorts pass into the niches below. Flues, *l*, fig. 151, 153, pass along the top of the furnace at each side, communicating with the stack, *e*, at the back. These are covered with flat, movable tiles, and communicate with the interior of the furnace by smaller flues, *m*. The course taken by the products of combustion in passing to the stack is indicated by the arrows.

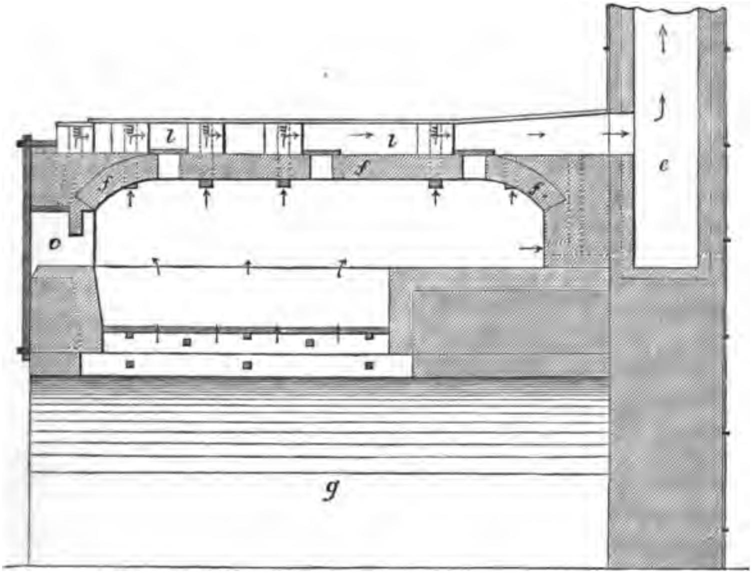

Fig. 151.—Silesian Zinc-Furnace, Llansamlet; longitudinal section.

This furnace is built of common brick lined with fire-brick, the whole being held together by iron bracings. In each arched recess are placed two retorts, A (fig. 152), the open spaces around them being filled in with pieces of brick, well plastered over with clay, to prevent the admission of air. During the working of a charge, each recess is closed by a movable sheet-iron plate, in which is a small peep-hole closed by a slide; the temperature is thus kept sufficiently high to prevent the solidification of zinc in the nozzles. The condensed zinc is conveyed by adapters into the recesses, *i*, where it is received in iron trays. The retorts or muffles, which are each about 3 feet in length, are made of a mixture of Stourbridge clay and the powder of old glass-pots, from which the vitreous matter has been removed previously to grinding. This mixture, suitably moistened, is tempered by working with a heavy rammer; and after being allowed to stand for a few days may be formed into retorts in moulds, capable of being divided longitudinally into pieces. The walls are made thicker towards the end furthest from the nozzle, as will be seen by reference to A, fig. 155. Each retort, after being air-dried during about three months, is strongly fired in an oven, heated either by a separate grate or by the waste heat from a distilling furnace, and is then ready for use. The opening, *n*, figs. 154, 155, serves for the introduction and drawing of charges. This, while the furnace is in operation, is kept closed by a luted tile. On each side of the mouth of the retort is a projection, *o*, supporting a bridge-piece, on which rests the clay nozzle, *p*, which is provided with an opening, *q*, closed during the working of the apparatus by a luted door, through which an iron rod may be introduced for the purpose of clearing the nozzle from condensed zinc. To the clay nozzle, *p*, a pipe, *r*, is fitted, and to this is attached a wrought-iron adapter, *s*.

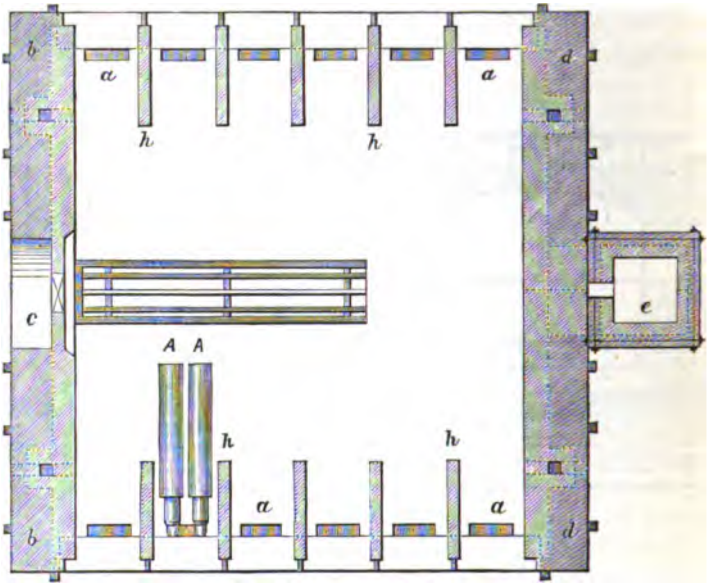

Fig 152.—Silesian Zinc-Furnace, Llansamlet; horizontal section.

Instead of the nozzles and adapters above described, the retorts used for the distillation of zinc by the Silesian process are now generally furnished with condensers of fire-clay and drums of sheet-iron similar to those employed for the pots of the Belgian furnace.

A new furnace requires first to be air-dried for several days, and then heated by a gentle fire; after some time the muffles are placed in position, their sides being protected from the flame by temporary walls. All openings are now closed, and the heat is gradually raised, until, after seven or eight days, the furnace has become sufficiently hot for charging. The protecting walls are now removed, and the spaces between the muffles and the openings in the front are closed by pieces of brick plastered with clay. Three or four light charges are worked off before the retorts are in a condition to receive full ones; new retorts always absorb a considerable quantity of zinc.

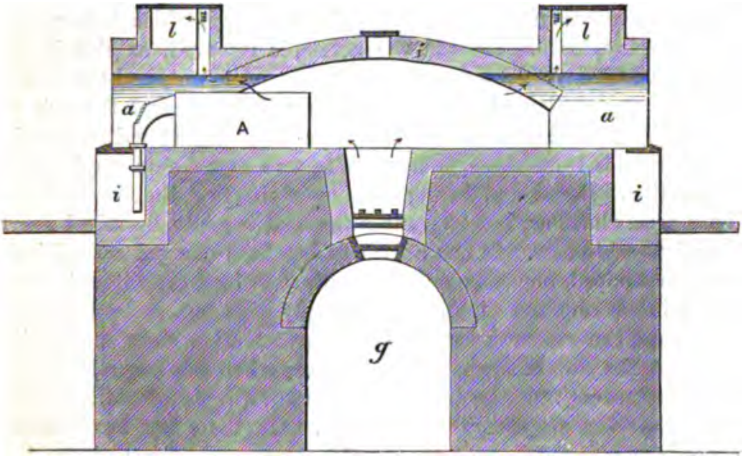

Fig. 153.—Silesian Zinc-Furnace, Llansamlet; transverse section.

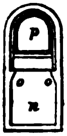

Fig. 154.

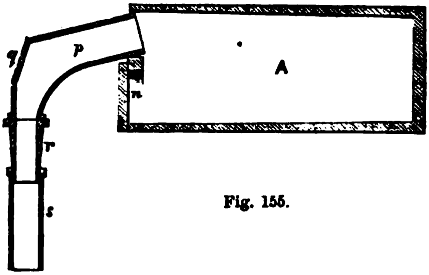

Fig. 155.

The ore treated in these furnaces is either blende or calamine; in either case it is previously calcined. When the former is used, it is in the state of a fine powder, whilst the latter is introduced into the retorts in pieces of nearly the size of peas. The charge for a furnace containing twenty-four retorts consists of about 1,500 lbs. of calcined ore, together with 100 to 150 lbs. of skimmings or dross, and about 7 cwts. of a mixture of coal and the fine cinders which fall through the bars of the grate. This mixture is not equally distributed amongst the retorts, those nearest to the grate receiving a somewhat larger amount than those further removed from it. Before charging a furnace which is in regular work, the clay door, *n*, beneath the nozzle, is removed, as also is that closing the orifice, *q*; the spent charge is then raked out, and the nozzles freed from any condensed zinc. The retorts are re-charged through the orifice, *n*, by the aid of a sheet-iron scoop; the cleaning and re-charging usually occupies from three to four hours, two men being employed in the operation. Before and during the time of charging, the fire is allowed to burn somewhat low, but afterwards the heat is gradually raised nearly to incipient whiteness. When a furnace has been charged and all leaks have been stopped, one man only is required to look after it and attend to the firing during the night. A uniformly high temperature must be kept up, and the condensers require frequent attention, since, should they get too hot, the zinc in them would take fire; when this occurs, the doors closing the recess must be taken down, and not replaced until the temperature has become sufficiently lowered. On the other hand, should the condensers get too cold the zinc in them solidifies, and must be removed by the introduction of a heated iron rod. A considerable loss of zinc is sometimes occasioned by cracks in the retorts; these must be immediately stopped by brushing them over with a mop previously dipped into a mixture of water and fire-clay.

The ore treated at Llansamlet was an argentiferous blende, very finely ground, and subjected to a careful roasting during twenty-four hours; the silver was extracted by a special process, and from the residue zinc was distilled.

The loss experienced during the calcination of the ore and the reduction of the zinc to the metallic state is said to have been unusually small; the greater portion of the zinc lost being retained in the residues. At the Llansamlet Works 11½ tons of coal were necessary for the production of 1 ton of zinc, at a cost for labour of about 2s. 8d. per cwt. of the metal obtained. This must, however, vary, since a given weight of zinc can be more cheaply extracted from a rich ore than from a poor one.

A furnace may be kept in continuous operation from twelve to thirteen months, at the expiration of which time it requires to be let out for repairs. Every four years each furnace must be taken down as far as the grate, and rebuilt.

The zinc obtained by this process required to be re-melted before it was ready for the market. It was fused in an iron pot set in brickwork over a properly constructed fire-place; this pot usually contained about 8 cwts. of metal, which was melted down in from half to three-quarters of an hour, the surface being covered with a mixture of oxide of zinc and finely divided metallic zinc. This was skimmed off and added to the next charge to be again subjected to distillation. The zinc obtained was tolerably free from impurities, with the exception of lead, which may to a considerable extent be separated by allowing the fused metal to cool nearly to its solidifying point before being laded into moulds; the lead, on account of its higher specific gravity, thus falls to the bottom, and the greater portion of it will be found in the last ingot. The ingots containing lead were put aside until a sufficient number had accumulated, when they were re-melted and the heavier metal allowed to settle. In Silesia the crude zinc is melted in clay vessels, in order that it may not be contaminated by contact with iron.

Modern Silesian zinc-furnaces are frequently heated by gas and provided with regenerators in accordance with the Siemens system. In such cases the brickwork of the regenerators is built unusually open to allow for the deposit of zinc oxide, which always takes place in them srom the breaking of retorts. The newer furnaces likewise often contain two series of retorts placed one above the other.

The ore is less finely ground than for the Belgian process, and is mixed with only one-third its weight of coal-dust. The operation usually lasts twenty-four hours, but sometimes extends over a longer period. With the old system of heating by coal the retorts lasted from forty to forty-five days, but in furnaces for which gas is employed their average duration is seventy-five days.

The cost of treating 1 ton of roasted ore may be reckoned at 75 f. 50 c., but the loss of metal is said to be slightly greater than with the Belgian furnace. The Silesian process is specially adapted for the treatment of poor zinc carbonates.

A large portion of the zinc annually produced is employed in the form of thin sheets. For this purpose it is necessary to re-melt the ingots, obtained by the treatment of zinc ores as above described. This is effected in a reverberatory furnace with an elliptical hearth, having a slight inclination towards one side. At the lowest point of the hearth is a hemispherical reservoir in which the fused metal is collected; the ingots to be re-melted are introduced through one of the doors and piled near the fire-bridge on the highest part of the hearth. The fused zinc is dipped out of this in iron ladles, and is poured into moulds which give it the form of slabs or plates suitable for rolling. These plates are subsequently re-heated in a second furnace built in the same mass of brickwork as the first, and heated by the waste gases escaping from it. Here they are elevated to a temperature not much exceeding 100° C., at which the metal is most malleable, and are then passed through the rolling-mill, by which they are reduced to the form of sheets.

## SODIUM.

Sodium was first isolated by Davy in the year 1807. It has a high lustre and a silvery-white colour, but light which has been repeatedly reflected from bright surfaces of this metal acquires a distinctly rosy tint. Its specific gravity at 15° C. is 0·985. Sodium is soft at ordinary temperatures, melts at 97·6° C., and oxidizes in the air. When placed on the surface of cold water it rapidly decomposes that liquid, but seldom takes fire unless the movements of its fragments are in some way restrained. With hot water it instantly takes fire, burns with a bright yellow flame, and a solution of soda is the result. Sodium is an abundant and very widely diffused metal occurring in vast quantities, as chloride, both in rock-salt and in sea-water, besides being present in salt-lakes, and in the waters of numerous mineral springs. It also occurs, though less plentifully, in various natural deposits in the form of sodium borate, sodium sulphate, and sodium nitrate. Glauberite, a natural double sulphate of sodium and calcium, forms extensive deposits in some parts of Spain and elsewhere, while many common and abundant minerals contain sodium as one of their essential constituents; it is present in albite, oligoclase, and various other felspars, as well as in natrolite, analcime, harmotome, and other zeolites. Sodium salts, in small quantities, are present in coal, and there are but few rocks, either eruptive or sedimentary, which do not contain traces of this element.

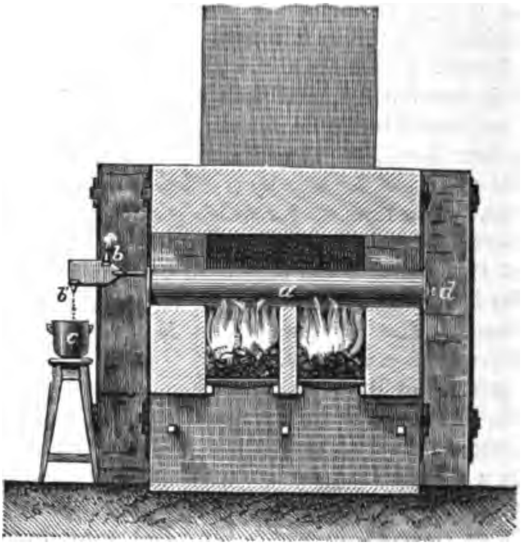

Fig. 156.—Preparation of Sodium.

## METALLURGY OF SODIUM.

By Deville's process, by which the most satisfactory commercial results are obtained, the preparation of sodium is conducted in an apparatus of which fig. 156 is a longitudinal section, partly in elevation. The retort, *a*, is a cast- or wrought-iron tube 3 feet 9 inches in length, having an internal diameter of 5½ inches and a thickness of ½ inch, covered exteriorly with clay. This is charged with a thoroughly incorporated mixture of 100 parts dry sodium carbonate, 45 parts of finely-ground coal, and 25 parts of chalk.

The receiver is a small cast-iron box opening along its centre and kept together by wrought-iron clamps; this is placed vertically, so that while the uncondensed products of distillation escape by the opening, *b*, into the air, the condensed sodium flows through the aperture, *b'*, and falls into a vessel, *c*, containing naphtha oil.[1] The charge is introduced at *d*, which can be closed by a luted stopper, in the form of cartridges made up in thick brown paper, so that when exhausted another charge may replace it without delay. The mixture employed should on an average yield from 28 to 30 per cent of sodium.

If the process has been conducted satisfactorily, the metal produced is almost chemically pure, but nevertheless small quantities of somewhat impure sodium always remain adhering to the surfaces of the receivers. When, therefore, a receiver has been once used, it should be opened and well scraped before being again employed. The impure metal thus scraped from the receivers is collected under naphtha and added to a subsequent charge of the retort. Lastly, the pure metal obtained by distillation is melted under naphtha, which is poured off as soon as the sodium is thoroughly liquid, and then run into moulds.

Sodium is principally employed in the preparation of aluminium, and an amalgam of this metal with mercury has been recommended for the treatment of certain gold ores.

*Potassium* is prepared nearly in the same way as sodium, but the retort employed is usually shorter than that used in the preparation of the latter metal, and has one end hermetically closed. The required heat is very considerable, and it is consequently found advantageous to sprinkle the retort, while still hot, with borax, in order to protect it from oxidation. The material operated on is either a mixture of charcoal and potassium carbonate, obtained by calcining argol in a close vessel, with the addition of a little chalk, or a mixture of 100 parts potassium carbonate, 20 parts of carbon, and 14 parts of chalk. When the retort has attained a red heat metallic vapours begin to be given off, and take fire when they come in contact with the air. The receiver is then put on, and while carbon monoxide burns at the end metallic potassium is condensed within. After the expiration of half an hour the receiver becomes filled with potassium, and is cooled in a hydrocarbon bath, while a fresh receiver is fitted to the retort.

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[1] Böttger recommends for this purpose a saturated solution of pure naphthaline in petroleum.

The crude potassium obtained is filtered by being wrung through a cloth saturated with naphtha, heated to 60° C., and the metal thus separated is re-distilled in a retort provided with a neck dipping into naphtha.

## ALUMINIUM.

Aluminium is a white metal, possessing a colour between that of silver and zinc. It is extremely malleable and ductile, and does not require frequent annealing during the operations of rolling or drawing into wire. Its specific gravity varies from 2·56, when obtained by fusion, to 2·67 after hammering. The melting point has not been determined exactly, but is estimated at about 600° C., between that of zinc and silver. Aluminium is slightly magnetic, and conducts electricity eight times better than iron. This metal, when pure, is not acted on by moist air, but slowly decomposes steam at a strong red heat; it is not attacked by cold nitric acid, and only slowly on boiling; dilute sulphuric acid has no effect upon it, but it is readily dissolved by hydrochloric acid with evolution of hydrogen. It is not attacked by sulphuretted hydrogen or even by sulphide of ammonium, and consequently preserves its colour in an atmosphere in which silver would rapidly become blackened. The vegetable acids exert no perceptible action on aluminium, and this metal is consequently well adapted for culinary vessels, particularly as any small quantities of alumina which might be formed would be perfectly innocuous. The hydrates of potassium and sodium, in a state of fusion, do not act upon aluminium, but their aqueous solutions dissolve it readily with formation of aluminate of potassium or of sodium and evolution of hydrogen. It may be fused without any kind of flux, and may be readily cast into any required form. When deposited from its solutions by electrolysis at low temperatures, it crystallizes in octahedra which appear to belong to the cubic system. Aluminium heated in a close vessel does not exhibit the slightest tendency to volatilize; it combines with sulphur at very high temperatures only, and is acted on but slightly by ammonia. Pure aluminium, even at high temperatures, is not sensibly affected either by air or by oxygen, and oxidizes very slowly, even before the flame of the oxy-hydrogen blowpipe. If, however, silicon be present, the metal burns rapidly, with formation of silicate of aluminium. Aluminium may, without oxidation, be fused under a layer of nitre, and from its physical and chemical properties may be regarded as occupying an intermediate position between the noble and the ordinary metals. Up to the present time aluminium has not found the extensive application for which its properties would appear to fit it. Its chief use is for the manufacture of aluminium bronze, and for making various ornamental objects.

Although minerals containing aluminium are perhaps more numerous and abundant than the ores of any other metal, they are mostly decomposable with great difficulty, and a very few only can be considered as ores of aluminium. These are—Corundum or emery, the commoner form of native aluminic oxide; bauxite, a natural aluminic hydrate; and cryolite, a double fluoride of sodium and aluminium, which is found in considerable quantity at Arksut-fiord in Greenland, and contains 13 per cent. of aluminium.

The estimation of aluminium is effected by the method already described for the separation of alumina in the analyses of iron ores. It is invariably weighed as $Al_2O_3$, which contains 53·2 per cent. of aluminium.

## METALLURGY OF ALUMINIUM.

This metal was first obtained in 1827 by Wöhler in the form of a grey powder, by the reduction of its chloride by potassium, and some years later he succeeded in preparing it in the form of small globules. The reduction of this metal has not hitherto been accomplished except by the aid of electrolysis, or of one of the metals potassium or sodium, which have a greater affinity for chlorine and fluorine than has the aluminium with which they are in the first instance combined. In practice, sodium is invariably employed to effect this decomposition, partly on account of its greater cheapness and steadiness of action, and partly because its lower atomic weight enables it to perform a larger amount of work than an equal weight of potassium.

Oersted first prepared aluminium chloride by exposing an intimate mixture of alumina and charcoal at a high temperature to the action of a current of chlorine. The chloride of aluminium formed, volatilized, and was collected in a suitable receiver; this he mixed with potassium amalgam, and on subjecting the mixture to heat, potassium chloride was formed, and aluminium combined with the mercury; this last was expelled by distillation, and a button of aluminium remained behind. The method of preparation at present adopted is in principle the same as that employed by Wöhler, and depends on the action of sodium at a red heat on the chloride or fluoride of aluminium, or, still better, on the double fluoride of aluminium and sodium.

We are largely indebted for our knowledge of aluminium to H. Sainte-Claire Deville, whose first researches were published in 1854. The process he originally employed consisted in passing the vapour of chloride of aluminium over metallic sodium maintained at a red heat in a tube either of copper or iron. Metallic aluminium was thus obtained, mixed with chlorides of aluminium and sodium, which were removed by washing with hot water. The globules of aluminium were consolidated by heating to the melting point and applying pressure. The metal thus obtained was subsequently re-melted and cast into bars.

The reduction of aluminium may be effected in an earthen or an iron crucible in the following way: 40 parts of the double chloride of aluminium and sodium, 20 parts of chloride of sodium, 20 parts of fluor-spar or cryolite, the latter being preferable, all finely pulverized and perfectly dry, are placed with from 7½ to 8 parts of sodium, in alternate layers, in the crucible, and the whole gradually heated until reaction begins to take place; the heat is now increased to bright redness, and the fused mass, after being well stirred with an iron rod, is poured into a mould. By this process four-fifths of the aluminium reduced may be obtained in a compact mass, and one-fifth in the form of divided globules enclosed in the resulting slag. The metal thus prepared is somewhat contaminated by silicon derived from the crucible, which is more or less attacked by the sodium and aluminium, as well as by the fluorides present; this may be, to some extent, obviated by lining it either with calcined alumina or with aluminate of calcium. When iron crucibles are employed, the aluminium produced is always found to contain a certain amount of that metal.

When large quantities are operated upon, the reduction is conducted in a reverberatory furnace; a mixture of 10 parts of the double chloride of aluminium and sodium and 5 parts of either fluor-spar or cryolite is heated with 2 parts of metallic sodium. The double chloride and the cryolite, or fluor-spar, are mixed in the state of fine powder with sodium in small ingots, and the whole is charged upon the bed of the furnace, which has been previously heated to the required temperature. An intense reaction takes place, a large amount of heat being developed, and the complete liquefaction of the mass is effected. As soon as the reaction is complete, the furnace is tapped in the usual way and its contents run off into a receiver prepared for that purpose. The slag thus obtained separates into two layers, of which the upper consists, to a large extent, of common salt, while the lower one, which is less fusible, chiefly consists of fluoride of aluminium. On pulverizing and sifting the latter, an additional amount of aluminium is obtained in the form of flattened shot; the fluoride of aluminium may be subsequently employed for the preparation of alumina. This process, which was patented in France and in this country by MM. Rousseau Frères and M. Paul Morin, is advantageous, inasmuch as the reduced metal is but little contaminated by silicon. The mixture at present employed in the works at Salyndres in France consists of double chloride of aluminium and sodium 12 parts, cryolite 5 parts, and sodium 2 parts.

The double chloride is obtained from bauxite after calcination with sodium carbonate, extraction with water, and evaporation to dryness, which gives aluminate of soda ($3Na_2O.Al_2O_3$). This is converted by hydrochloric acid into alumina and sodium chloride, and finally into the double chloride by mixing the latter products with charcoal, and strongly heating them in a current of chlorine.

In preparing aluminium from cryolite ($Na_6Al_2F_{12}$ or $6NaF.Al_2F_6$), which occurs somewhat abundantly in West Greenland, the pulverized mineral is mixed with half its weight of common salt, and the mixture arranged in alternate layers with sodium in an earthen or iron crucible; the proportions employed being 2 parts of sodium to 5 of cryolite. A layer of pure cryolite is placed on top, and the whole is covered with common salt. The crucible and its contents are heated until complete fusion of the mass has been effected, when it is well stirred with an iron rod and set aside to cool. On breaking the pot, the aluminium will be found accumulated at the bottom in the form of large globules, which are collected and re-melted. This is the process as originally practised by Professor H. Rose in Berlin, and by Dr. Percy and Mr. A. Dick in this country; and subsequently conducted on a manufacturing scale by MM. Tissier, at Amfreville, near Rouen.

Aluminium unites readily with most of the metals; with *zinc* and *tin* it forms brittle alloys; with *iron* it unites in all proportions, forming alloys which are both hard and brittle. When iron is present to the amount of 7 to 8 per cent. its alloys crystallize in long needles. Alloyed with even a small proportion of *silver*, aluminium loses its malleability. An alloy of aluminium with 3 per cent. of silver is employed for casting ornamental articles; it possesses the colour and lustre of silver, and is not blackened by sulphuretted hydrogen. *Aluminium-bronze* is an alloy of aluminium and copper; it has the colour of gold, is extremely hard, and takes a fine polish. It is very malleable, and possesses a tenacity nearly equal to that of mild steel.

In Messrs. Cowles' process of making aluminium alloys a mixture of emery, carbon, and granulated copper is subjected to the current of a very powerful dynamo-machine in a horizontal fire-brick chamber, whose walls are protected by making a central core of the ore and coarsely powdered gas carbon, while the sides and bottom are packed with fine charcoal. The current is led through the central core by carbon electrodes connected with the ends of the chamber. When the furnace is charged it is covered with fire-clay slabs, with perforations for the escape of gases, and luted with clay at the joints. A few minutes after starting the dynamo, carbonic oxide flames stream out of the openings, and complete reduction is effected in about an hour. The product is a rich aluminium-copper compound, from which the ordinary alloy known as aluminium-bronze may be made by re-melting with the necessary amount of copper.

## MAGNESIUM.

Magnesium is a metal of a silvery-white colour, malleable, ductile, and readily filed and polished. In many of its physical and chemical characteristics it closely resembles zinc, near the boiling point of which it fuses, and at a somewhat higher temperature can be distilled in an atmosphere of hydrogen.

Its specific gravity is 1·74, it crystallizes in octahedra, and its hardness is about equal to that of calcite. At ordinary temperatures magnesium remains unaltered in dry air or oxygen, but in a moist atmosphere it quickly becomes covered with a film of hydrate of magnesium. Heated to redness in the air it takes fire and burns with an exceedingly brilliant bluish-white flame with production of flocculent magnesia. Heated in watery vapour it becomes ignited, decomposes cold water slowly, but caidulated water very rapidly. If thrown into hydrochloric acid it is acted on so energetically as to cause combustion, although strong sulphuric acid dissolves it slowly with evolution of sulphurous anhydride. When heated in chlorine gas or in the vapour of iodine it burns, as it does also, although with less facility, in the vapours of iodine and sulphur. Magnesium likewise unites with nitrogen, forming a transparent crystalline nitride which has a composition represented by the formula $Mg_3N_2$. Alkaline and ammoniacal solutions, when cold, have no action upon this metal.

The light of burning magnesium being exceedingly rich in chemical rays, is often used for the production of photographs of the interior of mines, caves, and similar places inaccessible to the sun's rays. For this purpose a lamp is employed which, by a clockwork arrangement, supplies a magnesium ribbon as fast as it is consumed in the focus of a concave silvered reflector.

Magnesium occurs most abundantly in serpentines and in dolomite, which is a double carbonate of calcium and magnesium, as well as in various complex hydrated and anhydrous silicates. Brucite, a natural hydrate of magnesium ($MgH_2O_2$), is found associated with serpentine in Shetland, Sweden, the Urals, &c. Among the most important magnesium minerals are olivine, augite, hornblende, talc, meerschaum, and the principal members of the mica group. Magnesium is also present in sea-water, and in the waters of numerous mineral springs, as well as forming, chiefly in the state of carbonate and phosphate, essential constituents of plants and animals.

### METALLURGY OF MAGNESIUM.

By the process of Deville and Caron, which is founded on the reaction of sodium upon chloride of magnesium at a red heat, magnesium may be prepared on a manufacturing scale. Chloride of magnesium is prepared by the method adopted by Liebig, which consists in evaporating together a solution of magnesium chloride and sal-ammoniac. The dried residue is heated in a crucible until the whole of the ammoniacal salt has been expelled and the anhydrous magnesium chloride has become fused. It is, however, desirable that the mixed chlorides should be treated in a crucible which has been previously heated, in order that the operation may occupy the least possible time. When fused and in a perfectly liquid state, the magnesium chloride is poured on a clean plate of polished iron, leaving behind in the crucible a skeleton of infusible magnesia.

Fluoride of calcium is employed as a flux, but this is purified before being used. Crystallized fluor-spar is finely powdered in a mortar, and subsequently moistened with strong sulphuric acid, which, after the expiration of several days, is removed by repeated washings with distilled water. Silica and phosphoric acid are thus removed, while the residue, after having been heated to incipient redness, is ready for use.

To obtain magnesium, 150 parts of the anhydrous chloride of that metal, 120 of purified fluor-spar, and 56 of sodium, dried and cut into cubes of ⅜ to ½ inch, should be intimately mixed. This mixture is placed in a red-hot clay crucible surrounded by incandescent coke in an ordinary wind-furnace, and, as soon as it has been introduced, the pot must be closed by a cover, kept in its place by a red-hot fire-brick, as long as the reaction continues. When this, which occupies a few minutes only, has terminated, the crucible contains a liquid mass, consisting of a mixture of common salt, magnesium chloride, and calcium fluoride, through which are disseminated numerous globules of magnesium. This is briskly stirred with a stout iron rod, and the crucible, which is now removed from the fire, is allowed to cool during a few minutes. When sufficiently cold, small quantities of dry pulverized calcium fluoride are successively added, while the stirring of the mass, which gradually becomes pasty, is continued. By these means the small globules of magnesium are united into larger ones, and if the operation has been skilfully conducted, they finally become nearly all collected in a large button, which, on the cooling of the mass, is found occupying the upper portion of the saline scoriæ. Finally the crucible is broken, and the large button as well as a few smaller ones separated from the slags. The magnesium obtained should weigh about one-sixth of the weight of the anhydrous magnesium chloride employed, or about three-fourths of the total theoretical yield.

Instead of the before-mentioned mixture the following may be substituted, namely, 600 parts of anhydrous chloride of magnesium, 100 of Wöhler's flux, consisting of a mixture of the chlorides of sodium and potassium in equivalent proportions, 100 of pure fluoride of calcium, and 100 of sodium.

This is placed in a red-hot crucible, and the operation conducted as before described, except that fluoride of calcium is not added during cooling.

To convert the metallic magnesium into ingots, the buttons, after being cleaned by nitric acid, must be introduced into a hot crucible with a quantity of Wöhler's flux, and the whole rapidly fused. At first the metal floats on the surface of the slags, but at the moment of their solidifying the magnesium becomes more dense than they, and sinking to the bottom of the crucible is united in one button. The metal thus prepared is impure, containing carbon, silicon, and nitride of magnesium.

In order to remove these impurities it must be distilled in a tube made of gas-retort carbon enclosed in another of refractory clay well glazed on the inside. The space between these tubes, which should not be too great, is filled with fine sand, and is luted at the two ends with clay. The tubes thus arranged are enclosed in an inclined position across a small furnace of refractory clay, with the two ends projecting rather more than six inches on either side. The ends are stopped with corks carrying suitable glass-tubes, hydrogen entering by the upper end, and escaping by the lower one. In the centre of this carbon-lined tube the crude magnesium to be distilled is placed in a vessel hollowed out of a piece of retort carbon; hydrogen is introduced, and the tube is heated to incipient whiteness. The distillation of the metal, which is condensed in the lower portion of the tube projecting beyond the furnace, is indicated by the brilliancy of the flame of hydrogen issuing from the escape tube.

Grätzel's method, which has been in use for about two years in Berlin, depends upon the electrolysis of carnallite ($KCl.MgCl_2.6H_2O$, containing 34·2 per cent. of magnesium chloride). This, when perfectly dehydrated and melted in a crucible, is subjected for about thirty-six hours to the action of a current of moderately high tension from a dynamo, one of the terminals being connected with the crucible, while the other is plunged into the molten salt. The reduced magnesium separates in globules through the carnallite flux, and when collected and re-melted gives a chemically pure metal, which is sold in various forms, but more particularly as a fine powder for fire-work making. A small addition of magnesium to melted nickel is found useful in producing sound castings.

## MERCURY.

Mercury, or quicksilver, differs from other metals in being liquid at ordinary temperatures. It has a silver-white colour, with a strong metallic lustre, and when quite pure is not tarnished by exposure in the cold to a moist atmosphere. If, however, other metals are present, oxidation is rapidly set up, and the surface of the bath becomes covered by a grey powder. This metal is solid at a temperature of – 40° C., and is then both ductile and malleable. In high latitudes the cold is sometimes so intense as to cause the congelation of mercury; a similar result may be readily obtained by a freezing mixture of ether and solid carbonic anhydride. The same effect is also produced by a mixture of pounded ice and crystallized calcium chloride. If a rather large quantity of mercury be placed in a platinum dish, and gradually exposed to a proper refrigerating mixture, distinct crystals are obtained. The mercury in this case becomes solid around the sides of the vessel, and on pouring out the portion which still retains its liquidity, brilliant crystals, belonging to the cubic system, are found coating its sides. Considerable contraction takes place at the moment of congelation; for while its density at 0° is 13·59, that of frozen mercury is 14·4. Mercury boils at 350° C., and is distinctly volatile at all temperatures above 19° C. The latter property is readily shown by suspending a sheet of gold leaf in the upper part of a bottle in the bottom of which a little of this metal has been placed. On removing this arrangement to a cool place, and allowing it to remain a few days without being disturbed, that part of the gold which is nearest the surface of the mercury will be found to have become whitened by its vapour, while that portion of the sheet which is in the highest part of the bottle remains unaltered.

The mercury of commerce, when it comes directly from the furnace, is in most instances nearly pure, but is sometimes contaminated by holding small quantities of other metals in solution. With a view to the separation of these impurities mercury is often distilled from an iron retort, and again condensed in a vessel containing cold water. For this purpose one of the wrought-iron bottles in which quicksilver is imported may be conveniently employed. One of these, after being about half filled with the metal, should have screwed to it a piece of iron gas-pipe bent nearly at right angles, and furnished at its outer extremity with a covering formed of several layers of linen or cotton cloth, the end of which is made to dip into a basin of cold water. The open extremity of the iron pipe, together with the piece of linen hose attached, is moistened by a constant stream of cold water, which is made to flow upon them through a small stop-cock, and the iron bottle is heated in a furnace until vapour of mercury begins to be plentifully given off. The ebullition of the metal is sometimes attended with explosions, and care must be taken to so regulate the heat as to prevent the projection of any part of the charge through the iron tube into the receiver. By operating in this way, the greater portion of the foreign metals will be retained in the retort, whilst the mercury passes over in a purified state into the vessel containing cold water. A certain portion of the impurities is, however, generally carried over into the receiver; and consequently, when a pure specimen is required, their separation should be effected by some other means. The best method of doing this is to treat the mercury to be purified with nitric acid diluted with twice its volume of distilled water. The whole is then heated to about 50° C., and mercuric nitrate will be formed. The more oxidizable foreign metals present react on the latter salt, and are removed in solution as nitrates. Any oxide of mercury present is also dissolved by the nitric acid, with formation of mercuric nitrate. The action is continued during twenty-four hours, and the mixture occasionally agitated.

Lastly, the water is separated by decantation, and the nitrate obtained in a crystalline form by concentrating the solution. The metallic mercury, after being washed with distilled water, is dried first with bibulous paper, and subsequently by exposure under a bell-glass to the desiccating influence of caustic lime.

When mercury is merely soiled by a slight admixture of oxide, it may be cleaned by brisk agitation in a glass bottle with a small quantity of strong sulphuric acid. By this treatment the metal is divided into extremely small globules, which expose a large surface to the action of the acid. At the expiration of from three to four days the acid may be poured off, and the purified mercury washed and dried.

Mercury is not attacked by strong hydrochloric acid, even when its action is aided by ebullition. Dilute sulphuric acid likewise fails to dissolve it; but if concentrated acid be employed, it is, with the aid of heat, rapidly converted into mercuric sulphate, and sulphurous anhydride is evolved. Nitric acid attacks this metal with great energy, even in the cold, and when moderately diluted with water, and nitric oxide is plentifully evolved.

Mercury, both in the liquid and gaseous condition, combines readily with many other metals, such as gold, silver, zinc, tin, lead, and bismuth, and even dissolves them in small proportions without losing its fluidity. Such mercurial alloys, which may be separated from excess of mercury and obtained in a solid state by filtration combined with pressure, are called *amalgams*. This property is extensively applied in certain methods of extracting gold and silver from their ores, as well as in the method of *silvering* mirrors with tin. A dry tin and zinc amalgam forms the best exciter that can be applied to the rubbers of electrical machines. The great density and perfect fluidity of pure mercury, together with its regular rate of expansion by heat and its high electric conductivity, render it of immense value for the construction of physical instruments, such as barometers and thermometers.

Large quantities of mercury, both in the metallic state and in the form of various compounds, more particularly the oxide, chloride, iodide, and nitrate, are used in medicine; and there is also a very important consumption in the preparation of the artificial sulphide or vermilion, which is the most brilliant and permanent red colour known. Mercurous nitrate is sometimes used as a wash for hare and rabbit skins, and gives the fur the property of readily felting. The bichloride or *corrosive sublimate* is similarly used by taxidermists for preserving natural-history specimens.

## MERCURY ORES.

NATIVE QUICKSILVER; *Mercure natif; Gediegen Quecksilber.* Cubic.—Occurs in most mines producing mercurial ores. It is usually disseminated in small globules, but sometimes becomes so accumulated in cavities as to admit of being collected in considerable quantities.

CINNABAR; SULPHIDE OF MERCURY; *Cinabre; Zinnober.* Hexagonal.—This substance, which, practically speaking, is the only ore of mercury, crystallizes in tabular prisms, but commonly occurs massive, and is distinguished by its bright-red colour and scarlet streak. Sp. gr. 8·20. Its colour is crimson, scarlet, brownish, or orange-red; lustre semi-metallic, or adamantine. It is sectile, and in most instances nearly opaque.

Its composition is expressed by the formula $HgS$, corresponding to mercury 86·2 and sulphur 13·8 per cent. Cinnabar mostly occurs in connection with sandstones, talcose, carbonaceous, and argillaceous schists, or in limestones; it is not found in large quantities in crystalline or igneous rocks. When the ores of mercury are met with in stratified rocks they are often found in the form of veins or lodes; but when, as is frequently the case, the matrix is sandstone, they are commonly disseminated in minute grains through the mass.

*Metacinnabar,* a black amorphous sulphide of mercury of the same composition as but of lower density than the crystallized variety (7·7), is found associated with cinnabar in some Californian and Mexican localities.

Mercurial fahlerz or tetrahedrite, found at a few localities in Tyrol and Hungary, contains mercury up to a maximum of about 17½ per cent.

NATIVE CALOMEL; *Mercure chloruré; Quecksilber-Hornerz.* Tetragonal.—Colour white; lustre adamantine. Sp. gr. 6·48. Occurs at Moschellandsberg, in the Palatinate, coating the cavities in a ferruginous gangue, associated with cinnabar; also at the quicksilver mines of Idria in Carniola, at Almaden in Spain, &c. Formula, $HgCl$. Chlorine, 15·10; mercury, 84·90.

*Coccinite,* found sparingly in Mexico, is an iodide of mercury.

*Onofrite.*—A sulpho-selenide of mercury from San Onofre, Mexico.

*Ammiolite* contains antimonate of copper and sulphide of mercury. A specimen of this mineral from Chili, analysed by Rivot, contained 14·8 per cent. of tellurium.

### DISTRIBUTION OF MERCURY ORES.

The ores of mercury are very unequally distributed, being confined to a comparatively small number of localities, in some of which they occur in considerable abundance; nearly the whole of the metal furnished to commerce is consequently supplied by but a few mines. The geological range of these ores is exceedingly wide, as they are found in ancient as well as in the most modern formations. Some of the principal European mines of this metal are worked in rocks of Silurian age, while those enclosing the celebrated deposits of New Almaden in California belong to the Cretaceous period. In proof of the very modern origin of certain deposits of cinnabar, it may be stated that the fumaroles, at the Sulphur Bank near Clear Lake, in California, are at the present time depositing sulphide of mercury, together with various other minerals.

The vapours and gases issuing from the different crevices appear to be the agents by which the minerals now in course of deposition are brought to the surface. Sulphur is being condensed on the sides of many of the fissures, either in crystalline groups, as stalactites, or as translucent amorphous masses. This substance is sometimes intimately mixed with cinnabar, and more frequently with minute crystals of iron pyrites; or with pulverulent silica, often blackened by a tarry hydrocarbon. On the sides of many of the cavities gelatinous silica is deposited, coating chalcedony and opalescent silica in various stages of formation, varying from the gelatinous state to that of the hardest opal. This silica is sometimes colourless, but is more frequently permeated by cinnabar and iron pyrites, or is blackened by the tarry matter before mentioned. When the bituminous matter occurs in large quantity cinnabar is often replaced by globules of metallic mercury. Some of the silica intermixed with specimens from this deposit was, when first broken, so soft as to easily receive the impression of the finger-nail, but on reaching England it had assumed the hardness and appearance of ordinary chalcedony.

An analysis (in duplicate), made in 1873, of a specimen of this deposit, brought by the writer from Clear Lake in 1866, afforded the following results:—

| | 1. | 2. |
|---|---|---|
| $H_2O$ combined . . | 11·85 | 11·82 |
| $H_2O$ hygroscopic . . | 7·13 | 7·12 |
| $SiO_2$[1] . . . . . | 38·71 | 38·53 |
| $Al_2O_3$ . . . . . | 1·50 | 1·46 |
| $CaSO_4$ . . . . . | 2·56 | 2·84 |
| Fe . . . . . . | 9·25 | 9·34 |
| Hg . . . . . . | 7·08 | 7·08 |
| S combined . . . | 10·21 | 10·39 |
| S free[2] . . . . | 11·71 | 11·42 |
| | 100·00 | 100·00 |

The Sulphur Bank has been extensively worked as a quicksilver mine down into rocks of Cretaceous age.

Ores of mercury have been found in several localities in France, but they do not occur in sufficient quantities to admit of being profitably worked. The mercury mines of Rhenish Bavaria were formerly of considerable importance, and up to the beginning of the seventeenth century afforded from 150,000 to 180,000 lbs. per annum, but have now almost ceased to be worked.

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[1] Gives, with the combined water, the formula, $SiO_2.H_2O$.

[2] Dissolved out by carbon disulphide.

At Idria, in Carniola, mines of mercury have been in operation for several centuries, and are still of some importance; the ore, which is cinnabar, associated with a little native quicksilver, is disseminated in shale and black limestone of Triassic age. The yield of quicksilver from these mines was in 1877, 837,960 lbs.; in 1878, 791,015 lbs.; and in 1882, 836,000 lbs.

Bohemia formerly produced mercury in small quantities, and Hungary still furnishes a small amount. Cinnabar has been recently discovered in Servia.

The mercury mines of Spain are the most important in Europe, the principal workings being in the neighbourhood of the town of Almaden, in La Mancha, between Badajoz and Ciudad Real, in the centre of a series of deposits about ten miles in length, extending from Chillon to Almadenejos. Pliny asserts that the Greeks procured vermilion from the mines of this district 700 years before the Christian era, and according to the same authority they annually yielded to the Romans 100,000 lbs. of cinnabar. The mines of Almaden, now above 1,000 feet in depth, are opened upon three nearly vertical beds of granular quartzite or sandstone, associated with Silurian schists and an altered rock locally called *fraylesca*, which forms a belt between the stratified and the eruptive rocks. Two of these beds are each about 10 feet and the third 20 feet in average thickness; they follow the flexures of the enclosing rocks, and in one place come together, forming a mass about 45 feet wide. The cinnabar is diffused with considerable uniformity through the quartzite matrix, giving it the appearance of a bright red rock. The average yield of the ore is from 7 to 10 per cent., but on account of the nature of the metallurgical processes employed the loss of mercury is large. The average annual yield of quicksilver at Almaden during the last ten years has been 36,000 flasks, or about 1,242 tons. Since the year 1645 the mines have been worked on account of the government.

At Ripa, in Tuscany, mercury is obtained from several veins traversing mica-slate; the quantity annually produced is, however, small.

Ores of this metal are found in numerous places in Peru, and were known to the inhabitants long before the invasion of that country by the Spaniards; the mines of the province of Huancavelica, both as regards numbers and richness, are the most important. Those of Santa Barbara have been worked since 1566; but although at one time productive, their annual yield is now very small. According to Humboldt, this mine yielded, from 1570 to 1782, 1,040,452 quintals,[1] or about 43,000 tons of mercury; the average annual yield was less than 6,000 quintals; but in the best years it sometimes reached 10,500 quintals. From 1790 to 1845 the total yield amounted only to about 66,000 quintals.

Various other mines of this metal are worked in different parts of

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[1] Each about 100 lbs.

Peru, but they are of less importance and extent than those of Huancavelica; the total yield of the country was, according to Whitney, in 1854, about 203,000 lbs., of which one-half came from the mine of Santa Barbara.

Although immense quantities of mercury are consumed in Mexico for the process of patio amalgamation, almost the whole of that so employed is imported from other countries. At the beginning of the present century the annual consumption of mercury in Mexico amounted to 16,000 quintals. This was furnished by the Spanish government, who retained the sole right of supplying that metal, which was chiefly derived from the mines of Almaden and Huancavelica. At the time of Humboldt's visit, about the commencement of the present century, only two mines producing mercury were being worked in the country; one called the Lomo del Toro, and the other Nuestra Señora de los Dolores, yielding together only 70 to 80 lbs. per week. A mercury mine was wrought about the year 1844 near Guadalajara, but the results having been unsatisfactory, it was abandoned.

The existence of mercury in California was known, and works were established for the treatment of its ores, long prior to the first gold discoveries in that country. In 1845 a company was formed to work an extensive deposit of cinnabar at New Almaden, in one of the side valleys of San José; this has been a very productive and remunerative mine. The other mines which have been worked to any considerable extent are the New Idria, the Redington, Guadalupe, the Sulphur Bank, Pope Valley, and the San Juan. With regard to the geological position of the cinnabar deposits of California, it may be remarked that although this mineral has been found in formations of nearly every age, as far as is yet known there are no large and valuable deposits excepting in rocks belonging to the Cretaceous period.

The yield of the New Almaden mine from July 1850 to the end of 1883 was 793,859 flasks, or 60,730,213 lbs.; while its production during the latter year was 29,000 flasks, or 2,218,500 lbs. avoirdupois. The total production of mercury in California in the year 1883 was 46,725 flasks, or 3,574,462 lbs. The present total production of mercury in the world may be estimated at about 6,500,000 lbs. avoirdupois.

## ASSAY OF MERCURY ORES.

DISTILLATION WITH QUICKLIME IN AN ATMOSPHERE OF HYDROGEN.—All ores containing mercury, whether in the metallic state, or as oxide, sulphide, selenide, chloride, or iodide, admit, after being reduced to a fine powder, of being assayed with considerable accuracy by distillation with caustic lime, in an atmosphere of hydrogen. This operation may be conducted as follows:—A tube of hard glass, *a b*, of the diameter employed for making organic analyses, is drawn out at one of its extremities, in the way shown in fig. 157, and in this part a bulb, B, is so blown as to be between two parts of the narrowed tubing. The contraction at *a* is loosely plugged with asbestos, so as to allow of a free circulation of gas, while it prevents any solid matter from being drawn into the elongation between *a* and B.

Powdered quicklime is afterwards introduced into the tube, and slightly consolidated by pressure with a piece of glass rod, care being at the same time taken that the aperture be not closed; a weighed quantity of the ore to be assayed is then mixed with caustic lime and deposited at *c*. When the mixture has been placed in the situation above indicated the remainder of the tube is filled with lime, and its end closed with a perforated cork, into which a piece of small glass tubing is accurately fitted. The prepared tube, after being tapped on the table to form a small space above the material, is placed in an ordinary gas combustion-furnace, and a current of dry hydrogen is introduced by the extremity, *b*. The part of the tube between *b* and *c* is first warmed, and the heat is progressively advanced in the direction of *a*. The mercurial ore is by this treatment decomposed, and the volatile metal being carried forward by the current of dry hydrogen, is condensed and collected in the bulb, B, which is carefully kept cool. A small quantity of watery vapour is condensed at the same time, but by a continued evolution of dry hydrogen this is ultimately carried off. At the close of the experiment, when the whole of the mercury has been condensed and the watery vapour has passed off, the tube on either side of the bulb, B, is cut with a sharp file, and the bulb itself weighed with the mercury it contains. The metal is then poured out, and any portions which may still adhere to the glass removed by washing, first with a little nitric acid and subsequently with distilled water. After being thoroughly dried, the bulb is again weighed, and by subtracting the weight of the empty glass from the weight first obtained the quantity of reduced mercury is ascertained. In conducting this experiment it is of importance that a large quantity of moisture should not be contained in the substance operated on, since by its condensation in the bulb, and subsequent evaporation, a sensible amount of mercury is carried off.

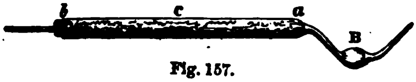

Fig. 157.

Distillation with Quicklime and Sodium Bicarbonate.—Instead of operating in the way above described, the following process may be adopted. An ordinary combustion-tube of about 18 inches in length, and closed at one end before the blowpipe, is filled for a short distance with sodium bicarbonate; this is pushed down to the closed end, so as to occupy a length of about 2 inches, and a little quicklime subsequently introduced. The ore to be assayed is intimately mixed with excess of quicklime, and then introduced into the tube, where it should occupy about 4 inches of the central portion; any particles which may have adhered to the mortar are removed by the aid of quicklime, and the lime which has been used for this purpose is likewise introduced into the tube. A layer of 6 inches of pure lime is placed upon this, and a loose stopper of asbestos is then pushed a distance of 4 or 5 inches down the tube; the anterior end of the tube being finally drawn out and bent at a somewhat obtuse angle. A few gentle taps on the table will suffice to shake together the contents of the tube in such a way as to leave a free passage throughout its whole length.

The tube, thus prepared and arranged, is introduced into a combustion-furnace, and its point placed in a receiving flask half full of water; the point is made to rest upon the surface of the water in the flask in such a way that its aperture may be partially closed. The tube is now slowly heated from the open to the closed end, as in the case of an organic analysis, and the last traces of mercurial vapour are finally expelled by heating the sodium bicarbonate, occupying a space of 2 inches, at its closed extremity.

While the tube still remains red hot the neck is cut off and carefully rinsed with a wash-bottle, transferring the rinsing-water to the receiving-flask; the small globules of mercury that have distilled over into the latter are then united into one large globule by agitation, and after the lapse of some time the perfectly clear water is decanted or drawn off by a syphon, and the mercury transferred to a weighed porcelain crucible, from which any adhering water is removed by blotting-paper. The mercury is finally dried over sulphuric acid under a bell jar, and the drying continued until its weight ceases to vary. This method, if carefully executed, yields accurate results; soda-lime may be employed in place of quicklime for mixing with the mercurial ore, but it does not offer any special advantage.

In the examination of very poor materials minute quantities of mercury may be collected by using the convex surface of a small gold dish filled with water as a condenser. The grey deposit so obtained is dissolved off the gold by nitric acid, and may then be tested by iodide of potassium or other reagents. Plates of gold are sometimes used in this way for determining the loss of mercury in the waste gases of mercury furnaces.

METHOD EMPLOYED AT IDRIA.—In establishments where many mercury assays have to be made daily, a small reverberatory furnace is employed. That used at Idria has, on one side, an iron plate with twenty-six holes for the insertion of a corresponding number of assays. Eight assays, each weighing four ounces, are taken from the charge of every furnace; these are mixed with two or three spoonfuls of powdered quicklime, and the several assays are introduced into eight iron retorts, and placed in the furnace. Receivers are then attached, the space between the two carefully luted, and the assays heated to bright redness; the distillation is finished when drops of mercury are no longer deposited in the receivers.

The distillation must be performed with a slowly increasing heat, and care taken that a sufficient temperature is attained by every part of the retort. Should the mercury obtained not unite well together, it is simply boiled in water for a few seconds. The last remnant of adhering water is removed by blotting-paper in the usual way, and after drying the mercury at a temperature of about 40° C., it is transferred to a tared watch-glass and weighed.

The assay is considered successful when the results of the various duplicates closely agree, and no undecomposed cinnabar is deposited in the receiver or in the neck of the retort. This method of assaying may be conducted with a sufficient degree of accuracy to serve either to control the operation of reduction, or for determining approximately the value of the ore; but the results obtained are in all cases somewhat below the truth. The lower, however, the amount of mercury contained in an ore, the greater will be the difference between the true percentage and the assay result; even with an ore containing 5 per cent. of mercury, the quantity found by this method will be lower than that which it actually contains. Various methods have been proposed for the volumetric assay of ores of mercury, but none of them admit of general use.

## METALLURGY OF MERCURY.

The extraction of mercury from cinnabar, which may be considered as, practically, the only ore of this metal, is effected either by the oxidation of the sulphur by the air, and the volatilization of the liberated metal, or by the use of re-agents, with which the sulphur enters into combination, while the liberated mercury distils over, and is condensed. When cinnabar is decomposed through the oxidation of its sulphur, the reactions which take place are expressed by the following equation: $HgS + O_2 = SO_2 + Hg$. When lime is present with cinnabar, and heat is applied, the following decompositions take place: $(HgS)_4 + (CaO)_4 = (CaS)_3 + CaSO_4 + Hg_4$. The choice of the method adopted for the metallurgical treatment of mercurial ores is chiefly influenced by their richness and quantity, the size of the fragments, and the local cost of fuel and materials. When the ores are rich and the quantities to be treated small, the gallery-furnace has sometimes been resorted to; but if the quantities are large, kilns are usually employed. Condensation is effected in large brickwork chambers, in clay tubes, in cast-iron pipes kept cool by water, and in chambers of wood and glass.

The following are the principal methods employed for the extraction of mercury from cinnabar:—

1. *Decomposition of the ore by roasting.*

*a.* Roasting in mounds; applied in Hungary to the treatment of cupriferous ores containing mercury.

*b.* Roasting in kilns or furnaces, carried on either by intermittent operations or continuously.

*c.* Roasting in reverberatory furnaces, and condensing in iron pipes; sometimes adopted for working schlich and other finely divided ores.

2. *Decomposition in retorts by lime.*

*a.* Gallery of the Palatinate.

*b.* Iron retorts.

## EXTRACTION OF MERCURY FROM CINNABAR BY ROASTING.

This method of extracting mercury from its ores is cheaper, and requires less time and fuel, than that by the use of lime in close vessels. On the other hand, mercury in the state of vapour becomes mixed with the products of combustion of the fuel used, and being carried forward by the draught, condensation is impeded, and loss of metal is the result.

At Idria, Almaden, New Almaden in California, and at Ripa in Tuscany, chambers, externally cooled by water, are employed as condensers, while at some other establishments they are constructed so as to allow jets of water to play within them. At the Pioneer Mine, in California, the condensers were traversed by troughs, through which a stream of water constantly flowed. The porous masonry of large chambers is, however, found to imbibe a very considerable quantity of mercury; and they have consequently been replaced by cast-iron pipes cooled externally by water, and connected with condensers provided with a suitable chimney. These tubes are, however, found to be acted upon somewhat rapidly by the sulphurous vapours passing through them, and have in consequence been, in some instances, replaced by wooden pipes. Wooden pipes possess the advantage of being more durable than iron ones, and can, by means of small doors, be easily opened and cleaned. The mercurial fume which collects in them is also free from oxide of iron, and is consequently more readily worked.

The treatment of mercurial ores on a large scale is still, to some extent, conducted in kilns working intermittently, which, after the exhaustion of each charge, require to be allowed to cool before it can be withdrawn and replaced by a fresh one. This circumstance not only causes loss of time, but from the alternate heating and cooling of the masonry the walls are liable to crack, and to allow of the escape of an amount of vapour resulting in a serious loss of that metal; at the same time most deleterious effects are produced by the mercurial fumes on the health of the workmen employed.

### *a.* ROASTING IN MOUNDS.

In some parts of Hungary copper ores are found containing from ½ to 17½ per cent. of mercury, but probably averaging from 2 to 2½ per cent. of that metal. These ores are roasted in heaps preparatory to their fusion for copper matte, and during the operation a portion of the mercury is collected. The mounds are usually about 40 feet in length, 20 in breadth, and 3 feet 6 inches in height, and are provided with flues and chimneys similar to those sometimes employed for the preparation of coke. Hot embers are thrown into the vertical shafts, which are subsequently filled with small coals; the decomposition of the sulphides commences, and mercury condenses in drops on the outer and colder surfaces of the ore. Before the uppermost layer becomes so far heated that mercury is re-volatilized, it is covered by a fresh stratum, as are also all places where any sinking of the covering has taken place. The process of roasting is complete in about three weeks, and the upper layers of the mound will then be found to contain mercury resulting from the condensation of mercurial vapours expelled from the more central portions of the pile. These upper layers are removed, and washed upon an iron sieve in a tub of water, in which the mercury and fine ore accumulate; while the coarser portions which remain on the sieve form the lower layers in the next mound prepared for calcination. The fine ore is subsequently washed on very close sieves, which allow the mercury to pass through them and to unite in the liquid state, while the fine ore remains behind and is added to the next mound for roasting. The upper layers are thus treated so long as they continue to yield an appreciable amount of mercury, and the lower ones, which have been completely roasted are taken to the furnace in which the first fusion for matte is effected.

By this method of treatment ores are stated to yield 75 per cent. of the mercury indicated by assay, and the collection of quicksilver from copper ores containing only ¼ per cent. of that metal is said to be attended with profit; in 1861 Hungary produced 31¼ tons of mercury from the working of copper ores of this description.

### *b.* ROASTING IN KILNS.

TREATMENT OF MERCURIAL ORES AT IDRIA. *Old Process.*—The ores treated were divided into two classes; mineral in lumps varying from the size of a nut to a cubic foot, and those fragments of which the size ranged from below that of a nut to the finest dust.

The first class comprised three subdivisions, namely, the poorest kind, affording only 1 per cent. of mercury; the massive sulphide, consisting of the richest selected fragments; and lastly, pieces, containing from 1 to 40 per cent., arising from the breaking and sorting of the different ores.

The second class was also subdivided into three varieties, and comprised fragments extracted from the mine in small pieces, and which, on an average, afforded from 10 to 12 per cent. of mercury; bits of ore separated by washing on a sieve, containing at times 30 per cent. of metal; and lastly, sand and *schlich* obtained by stamping and washing the poorer ores; this generally afforded a produce a little superior to 8 per cent.

The metallurgical treatment of these several products consisted in subjecting them to a process of roasting in a large kiln-like apparatus, in which the sulphur was converted into sulphurous anhydride, while metallic mercury was set free and condensed in a series of chambers arranged on either side for that purpose.

This apparatus consisted of two large roasting kilns, A, figs. 158, 159, furnished on either side, with a series of chambers, C, in which the mercurial vapours were condensed.

The larger fragments of the mineral treated were closely piled on the hollow arch, *a*, until the space between it and the next had been entirely filled. On the second perforated arch, *b*, were placed fragments of smaller dimensions; and on the third, *c*, were originally deposited, in earthen vessels, the slimes arising from the mechanical treatment of the poorer ores. This washed schlich was, however, subsequently formed into bricks, and charged into the furnace, mixed with lump cinnabar.

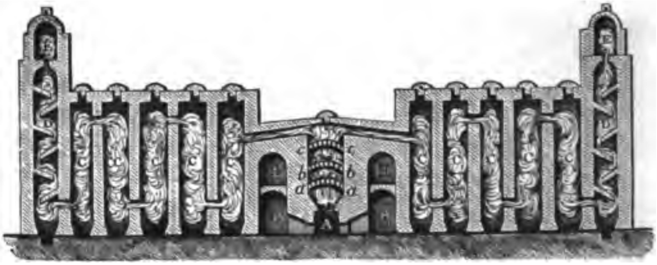

Fig. 158.—Old Furnace at Idria; longitudinal section.

When the furnace had been thus charged, a fire was lighted on the grate, and the heat progressively raised until the decomposition of the mineral began to take place. The cinnabar placed in immediate contact with a current of heated air, which entered the furnace through apertures opening into the spaces, G, H, was decomposed and the metal sublimed; the latter being carried into the condensing chambers, C. The greater portion of this mercury was condensed in the first three chambers, and was conducted by the gutters, *x*, *y*, *z*, to covered reservoirs beneath the level of the floor. In the last chambers of the series a considerable amount of water, and but little mercury, was condensed. These products were, on account of the impurities they contained, carried

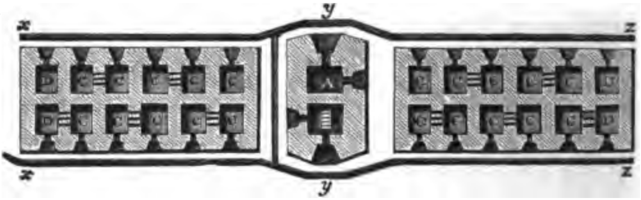

Fig. 159.—Old Furnace at Idria; horizontal section.

off by gutters, to a tank, in which they accumulated. The mercurial dust, associated with the metal obtained from the last chambers, was subsequently separated by filtration and mixed with some of the finer ores, to be again treated in the furnace.

In order to condense, as far as possible, the last traces of mercury passing through the apparatus, a stream of cold water was made to flow through the chambers, D, on inclined tables, extending nearly from one wall to the other, and between these the vapours and gases circulated before escaping, through E, into the air.

The mercury was afterwards filtered through thick canvas bags to separate solid impurities, and subsequently packed in wrought-iron bottles for exportation.

This arrangement, which was one of the largest single metallurgical structures in the world, was charged in three hours by the united labour of forty men. The wood employed as fuel was beech, and the distillation lasted from twelve to fifteen hours, during which time the whole interior of the kiln was kept at a cherry-red heat. A complete charge for the double apparatus was from 1,000 to 1,200 cwts. of ore, which produced from 80 to 90 cwts. of mercury. The furnace required, according to the season of the year, from four to five days to cool; and therefore, when the time necessary for charging and withdrawing the residue is included, only one distillation could be made in the course of a week. This furnace was 180 feet long and 30 feet in height, and was first erected at Idria in the year 1794, before which time an aludel-furnace, similar to those of Almaden, was employed.

*Continuous Process.*—The first furnaces employed for this process were constructed by Bergrath Hähner in 1850, and were used with great advantage for the treatment of all but the more finely divided varieties of ore; but even these could be advantageously treated if first mixed with clay and made into bricks. The furnace was a cylindrical kiln furnished at bottom with a movable grate constructed of iron bars, each of which could be separately withdrawn, and beneath which a small waggon could be placed for the purpose of removing the exhausted residues after calcination. This cylindrical kiln was fed at top by a hopper closed by a valve, and was connected, by a lateral flue immediately below it, with a series of six condensing chambers, built of masonry, and externally covered with iron plates, kept cool by a continuous flow of water. The chimney was built in tiers, each of which was cooled by water, while the chambers, which communicated with one another alternately at the bottom and top, had clay floors, tightly rammed in on a foundation of masonry.

When this arrangement was lighted, a few pieces of broken brick were placed on the bars so as to diminish the interstices between them, and on these were laid some brushwood and charcoal, on which the first layer of mercurial ore was charged. The kiln was now filled with ore and charcoal, in alternate layers, to a height of about 12 feet; the wood upon the grate was kindled, and the fire made its way slowly upwards through the mass. Charges of 7 cwts. of ore, with from 3 to 4 per cent. of charcoal, were let down through the hopper every forty-five minutes, while the exhausted ore was from time to time withdrawn by removing some of the bars at the bottom, and allowing it to fall into small iron waggons running on a narrow tramway. The ores remained twenty-two hours in the kiln, and those containing 3·11 per cent. of mercury by assay yielded 1·90 per cent. on the large scale by this method of treatment.

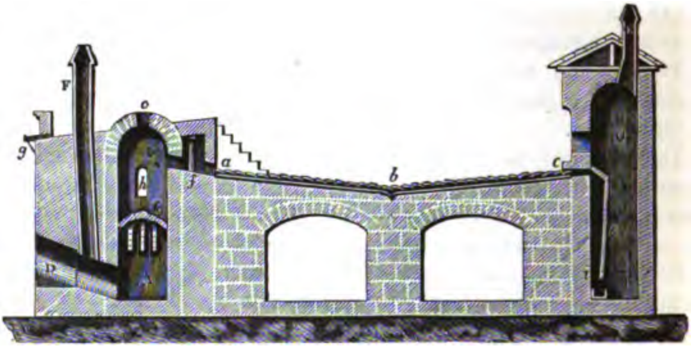

Fig. 160.—Aludel-Furnace; longitudinal section.

More recently an improved continuous coarse-ore furnace has been introduced at Idria by Bergrath Exeli, on the model of the old Rumford lime-kiln, from which excellent results are obtained. Figures and a description of a slightly modified form of this furnace, as employed for the production of mercury at New Almaden, California, will be found on p. 551.

Aludel-Furnace of Almaden.—This apparatus, figs. 160 and 161, of which the first is a vertical section and the second a sectional plan, was introduced in 1646 by Juan Alonzo Bustamente, from Huancavelica, in Peru.

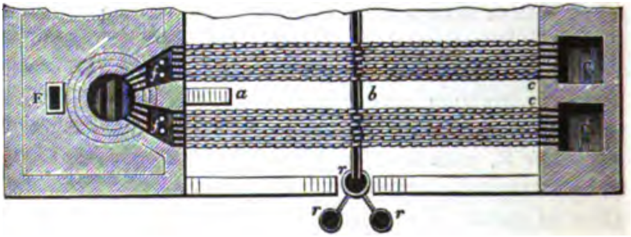

Fig. 161.—Aludel-Furnace; sectional plan.

These furnaces, which are called *buitrones*, consist of a circular kiln, A B, separated into two compartments by a brick arch, *k*, pierced with numerous apertures. The ore is piled in the space, B, above the arched diaphragm, the larger masses being placed first, and the smaller fragments upon them. The top is then covered with lumps formed of clay, kneaded with fine schlich. At the upper extremity of the cavity, B, is arranged a system of openings, *f*, which communicate with a series of earthen adapters, fitting one into another, and resting on the doubly inclined surface of the terrace, *a*, *b*, *c*. These earthen pipes, or *aludels*, fig. 162, are thrust one into another, and luted with a little softened clay, by which the leakage of the joint is practically obviated. Some of the condensed mercury remains in the aludels, but a portion flows through a hole pierced in the aludel at the lowest part of the series, and is collected in the gutter, *b*, by which it is conducted through wooden spouts into receiving basins, *r*. The uncondensed gases, mixed with mercurial vapours, pass through apertures, *c*, into chambers, C, where, passing under a diaphragm, *e*, a certain portion of the metal is deposited in a trough, *i*, filled with water. What still remains uncondensed passes into the upper part of the chamber, whence it escapes into the atmosphere through a chimney, E. The mercurial soot which accumulates on the sides of this chamber is occasionally swept down, and, after being kneaded into bricks, with the addition of clay, is again treated in a subsequent operation. The fuel employed is brushwood, which being ignited in the space, A, beneath the arched diaphragm, affords the heat necessary for working the furnace. The aludels are placed in twelve ranges of about twenty-five in each; the fuel is introduced through the opening, D, and the smoke and products of combustion are principally carried off by the chimney, F; *a* is a flight of steps for mounting on the top of the furnace, and *g* a gutter by which rain-water is carried off. The ore is introduced into the furnace through the door, *h*, and opening, *o*, which are afterwards securely luted. The firing is continued during twelve or fifteen hours, and the apparatus is then allowed to cool for three or four days, when the residues are removed and it is charged for another operation.

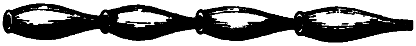

Fig. 162.—Aludels.

Furnaces similar to those used at Idria were introduced at Almaden about 1806, their capacity being two and a half times greater than that of the aludel-furnace.

New Almaden, California.[1]—The ores treated at the reduction works or *hacienda* belonging to this mine are divided into the four following classes:—

| | | | | | |
|---|---|---|---|---|---|
| *Granza*, | in fragments | varying from | 9 to 3½ | inches, | rich, 6 to 8 per cent. |
| *Terrero* | „ | „ | 6 to 3½ | „ | poor, 1 to 2 „ |
| *Granzita* | „ | „ | 3¼ to 1¼ | „ | poor, 1 to 3 „ |
| *Tierras* | „ | „ | 1¼ to dust | . | poor, 1 to 3 „ |

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[1] 'Quicksilver-Reduction at New Almaden,' by Samuel B. Christy, University of California. Transactions of the American Institute of Mining Engineers, 1885.

*Intermittent Furnace.*—The last survival of the old intermittent furnace consists of an ore-chamber 12 feet in length, 9 feet in width, and 17 feet 6 inches in height, inside measure, which is separated from a fire-place at one end, and from the vapour-chamber at the other, by a diaphragm of hollow brickwork. This is arched inwards in order better to withstand the pressure of the enclosed ore.

The mineral is charged into the ore-chamber through the top of the furnace, and a series of channels or flues is built longitudinally along the bottom with the larger lumps of ore, in continuation of the apertures in the pigeon-holed wall. When the first series of channels has been thus built, a layer of ore from 2 to 3 feet in thickness, according to the size of the fragments, is charged into the furnace; another series of channels is then constructed, and so on to the top of the furnace. To counteract the tendency of hot air to roast the upper rather than the lower layers of ore, the channels are made smaller and further apart in the upper part of the ore-chamber, while a certain amount of smalls, and of soot from the condensers, is added for the same reason.

There are four discharge-ports or draw-holes, two on each side of the furnace, which are bricked up during the roasting of a charge; while the flue for taking off the vapours has its bottom level with that of the ore-chamber in order to further counteract the upward tendency of the heated gases.

The operation of charging requires the labour of eight men for one day, the average weight of the charge being about 100 tons of mixed *granza* and *terrero.* When the furnace is charged the draw-holes are bricked up, and some pieces of old sheet-iron laid on top of the charge; this is covered by a layer of stable manure, that again by 3 inches of clay, and a wood fire is lighted upon the hearth. The roasting of the ore requires the labour of one man per shift of twelve hours for five days and four nights.

When the firing has been stopped the furnace is allowed to cool during three days and nights, the air being allowed to pass through the condensers the whole time.

For discharging the residues the labour of four men during one day is required. The top is removed to create an upward draught through the furnace, the discharge-ports are opened, and the spent ore is drawn out into slag-cars. In this way each charge of ore is worked in ten days, so that three charges may be put through the furnace in a month.

The ores treated in this furnace yield on an average 9·16 per cent. of mercury, the cost of working being $1·368 per ton (2,000 lbs.) of ore.

*Continuous Coarse-Ore Furnaces.*—These furnaces are, with the exception of some slight modification of detail, similar to the coarse-ore furnaces introduced at Idria by Bergrath Exeli, being small stack-furnaces fired from the outside.

Figs. 163, 164, 165, 166, represent, after Mr. Christy, sections, an elevation, and top view of this furnace. It will be observed (fig. 165) that the lower half is a regular hexagon, with abutments at the alternate sides containing the fire-places, ash-holes, and draw-pits. Each

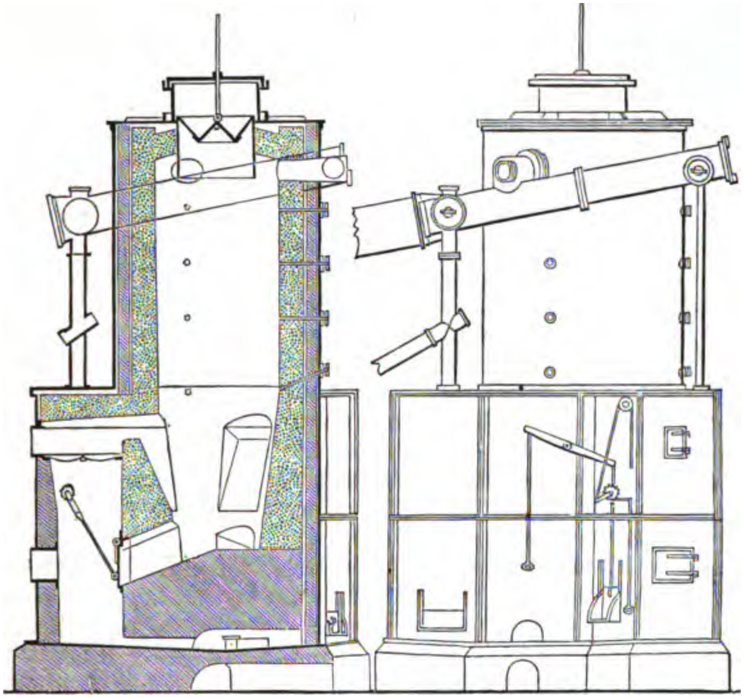

Fig. 163.—Section on A B. Fig. 164.—Elevation.

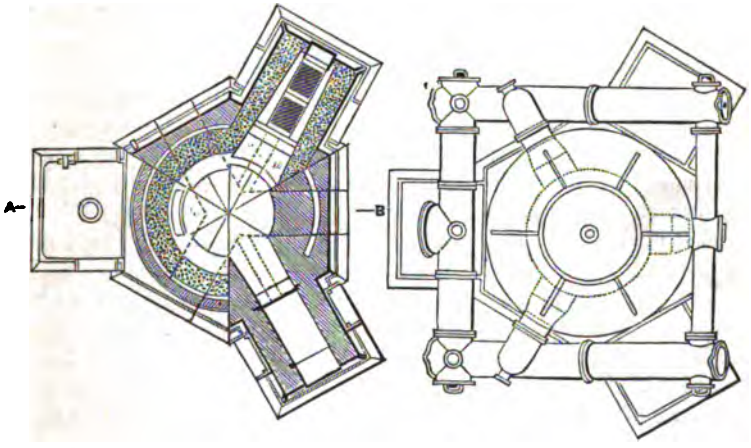

Fig. 165.—Horizontal Sections. Fig. 166.—Top View.

Continuous Coarse-Ore Furnace.

abutment has on its face a fire-door and an ash-pit door, the latter commanding the discharge-door for spent ore.

The residue or spent ore, after being drawn from the cooling-pits of the shaft into the ash-pits, is allowed to remain until its fumes are exhausted, and is then drawn into "slag-waggons" through the discharging-doors. These discharging-doors are placed on the side of each of the three abutments at a lower level than the ash-pit doors.

The upper half of this furnace is cylindrical, and is closed at top by a flattened dome supporting the charging apparatus in its centre. The fumes are removed from the vapour-chamber, above the surface of the ore, by cast-iron pipes 12 inches in diameter. In one of the two furnaces of this description employed at New Almaden there are three of these discharge-pipes placed at the top of the shaft on the sides opposite each fire-place. In the other there are, in addition to these, three pipes placed at the top of the shaft, one above each fire-place. By means of short iron pipes these are connected with the rectangular system of pipes encircling the head of the furnace, which, inclining downwards at an angle of 10°, lead to the condensers. In order to keep these pipes clear of soot each contains an iron scraper moved by a rod passing through a stuffing-box at the angles of the system of pipes. These are not often used, and at other times the rods are securely luted in their places with clay.

The cylindrical stack of this furnace has an interior diameter of 6 feet, with a height of 11 feet 6 inches, and is joined to the frustum of a cone 8 feet in depth, which contracts at the bottom to a diameter of 4 feet. Along the sides of the cylindrical part of the shaft, opposite each of the fire-places, is a series of peep-holes, which are usually closed gas-tight, but are occasionally used to determine the height and temperature of the column of ore.

The entire structure rests on a slightly concave iron plate placed on the top of the foundations, which causes any mercury that may permeate the masonry to flow into a reservoir prepared for its reception. Experience has, however, shown that this precaution is unnecessary.

The lower part of the furnace is enclosed by cast-iron plates bolted together and cemented with rust joints. The cylindrical portion has a jacket of sheet-iron, and is covered at top by a cast-iron plate supporting the charging apparatus, which renders the whole vapour-tight.

The charging-apparatus consists of a cup and cone, but, owing to the poisonous nature of quicksilver and its fumes, an additional cylinder, which can be closed by a cover provided with a water or sand joint, is also added. The rod moving the cone passes through a stuffing-box in the centre of the cover, and is in connection with a counterpoise weight. The cover itself is also attached to balance weights by two chains passing over pulleys. By the use of this contrivance the charging of the furnace is effected without exposing the workmen to the effects of mercurial fumes. The cover is lifted, a charge of ore mixed with a small proportion of fuel is tipped from an ore-waggon into the hopper, and the cover is lowered into its place. The ore is now allowed to remain in the hopper until it has acquired the temperature of that part of the apparatus, since if dropped into the shaft in a cold state it would be liable to chill the fumes and cause their condensation in the furnace.

At the proper time, an equal quantity of spent ore having been withdrawn into the ash-pits, the cone is lowered and a charge dropped into the furnace.

On starting a campaign with these furnaces they are filled to above the height of the fire-place with spent ore, and then with raw ore to just below the level of the uppermost peep-hole. As this height is never exceeded, there is always above the top of the ore a vapour-chamber in which the fumes collect before passing out through the iron pipes to the condensers. With the ore is mixed 1½ per cent. of charcoal, coal, or coke to assist in raising the temperature of the furnace, and, at the same time, to keep the column of ore more open. A good fire of oak or pine-wood is maintained in the three fire-places, and as soon as the lower peep-hole indicates a dull cherry-red heat, a quantity of spent ore is drawn into the ash-pits, and a charge, consisting of 1,600 lbs. of ore, with 24 lbs. of fuel, is dropped into the furnace from the hopper. The spent ore is allowed to remain some time in the ash-pits, so that it may discharge any remaining mercurial fumes through the fire-place into the furnace, and is afterwards drawn into slag-waggons through side doors in the abutments. This series of operations is repeated every two hours during the whole period the furnaces are in action.

Each furnace roasts about 9⅝ tons of ore in twenty-four hours, and as it holds 21 tons, it takes 52½ hours for a charge to pass through it.

The ore treated contains, on an average, 7 per cent. of quicksilver, and the total cost of working a ton of ore is $0·952.

*Continuous Fine-Ore (Granzita) Furnace.*—The question of discovering some cheap method of treating the large quantities of poor smalls produced at the New Almaden Mine has always been one of great importance. Making this ore into balls or bricks with clay would add at least 70 per cent. to its cost of treatment, while the reverberatory furnaces used at Idria would, on account of the high prices of labour and fuel in California, be equally out of the question.

This difficult problem has, however, been satisfactorily solved by the adoption of the Hüttner and Scott furnace, which may perhaps be regarded as the most important modern contribution to the metallurgy of quicksilver.

The general idea of this furnace is similar to that of the Hasenclever-Helbig shelf-furnace, in which a series of inclined shelves, placed on the opposite walls of a vertical shaft, retard the descent of a column of somewhat finely divided ore. It, however, differs from the Hasenclever type in combining a number of ore-chambers in the same mass of brick-work; as also in the devices for regulating the products of combustion and for effecting the discharge of the burnt residues. Several furnaces upon this principle have been erected at New Almaden, but that which has been selected for description is the last built and most efficient of these appliances.

In the folding plate facing p. 556, fig. 1 is a half front elevation and half vertical section of this furnace, and fig. 2 vertical sections through the fire-place and ore-chamber.

The vertical section through the fire-place, *a*, shows the ash-pit, *b*, and the second vapour-chamber, *e*, on that side of the furnace. Along the wall between the fire-chamber, *a*, the vapour-chamber, *e*, and the ore-chambers, are shown pigeon-holes, *c*, through which the flame and vapours from the ore pass into the ore-chamber and out of it again. The pigeon-holes in the walls of the fire-chamber are cut so as to slant with the shelves; while those in the vapour-chambers are cut horizontally, but are so bevelled that the ore from the ore-chamber and dust from the gases cannot obstruct them. The former falls back into the ore-chamber, while the latter settles in the vapour-chambers, whence it is removed at the end of the campaign through the openings, *g*, which are kept bricked up during the run.

In case any obstruction should occur either in the pigeon-holes or on the shelves, peep-holes, *x*, are placed in the axes of the pigeon-holes in the end walls of the furnace. These, which are placed on the same line opposite one another, under the extreme end of each shelf, are made of 3-inch gas-pipe, and are usually kept luted with clay. When any obstruction occurs this is easily removed by the introduction of an iron bar.

An earthenware pipe, *l*, having an internal diameter of 16 inches, passes through the condensers, thus cooling them and furnishing the fire-place with slightly heated air. Branching from it, and leading to the pigeon-holes, are the hot-air flues, *m*, which cool the wall; thus assisting to render the combustion perfect, and at the same time to prevent the deposit of soot.

The vertical section through the right hand double ore-chamber (fig. 2) shows the arrangement of shelving and the way in which the ore descends. The horizontal length of the ore-chambers of this furnace is 11 feet 6 inches, the width of each 25½ inches, and their height 27 feet 3 inches, from the roof to the point of discharge. The tile shelving, *s*, is placed at an angle of 45° with the side-walls; the shelves are 30 inches apart vertically, those on one wall being 15 inches above those on the other, and at right angles to them. They are made of ordinary fire-tiles 36″ × 16″ × 3″, four of which make a length, and allow of a 3-inch projection into the end walls. They are not let into the side walls, but are at top supported by slight abutments projecting from the wall, while at bottom they rest at intervals of 18 inches on ordinary fire-bricks.

The discharge is arranged by supporting the end of the ore columns of the interior chambers on the bottom of the draw-pit; the ore of the exterior chambers being supported by cast-iron plates. The latter have, in front, aprons which discharge the ore into the lower draw-pits, whence the spent material from both divisions is raked into waste-waggons.

The ore thus lies at its natural slope until it is raked out, when the whole column descends gradually from bottom to top. There are four double discharges, *j*, *k*, on each side of the furnaces, which are kept closed by sheet-iron doors to protect the workmen as much as possible from the hot dust. Inclined iron plates are placed in the foundation of the furnaces to prevent any waste of mercury.

In fig. 1 is shown a half elevation and a longitudinal half section, both fronts of the furnace being the same. From the fire-chamber, *a*, the heated gases pass through the lower third of the ore-chambers to the vapour-chamber, *d*, shown by dotted lines on the left. Thence they pass through the middle third of the ore-chambers to the vapour-chamber, *e*, on the right, whence they are conducted through the upper third of the ore-chambers to the vapour-chamber, *f*, on the left, and thence through the iron pipe, *i*, to the condensers.

The ores at New Almaden usually cause no difficulty from fusing or clotting, but sometimes in winter, when charged in a damp state, they are found to become baked on the upper shelves so as to require stirring, and to obviate this the rakes, *r*, have been introduced. One of these rakes extends through each of the ore-chambers just above the topmost shelves, where the ore enters the furnace from the hoppers, *h*. Should any baking of the charge take place, the lumps are readily broken by the use of the rake, and the ore descends regularly into the furnace.

This furnace holds 45 tons of ore, and 36 tons are passed through per twenty-four hours, and hence each charge remains in the apparatus during thirty hours.

The working of the furnace is conducted in the following way:—One ton by volume[1] of the spent ore is drawn alternately from each side of the furnace every forty minutes, 250 lbs. being taken from each draw-hole, and a charge of 2,000 lbs. of raw ore and 20 lbs. of coal is let down from the hopper, *h*, which is again immediately filled with raw ore mixed with its proper amount of fuel.

The firing is attended to every hour, three 4-feet lengths of oak or pine being ordinarily added to each side of the fire-place during that period, and from 1½ to 1¾ cord of wood is thus burned every twenty-four hours. The average amount of mercury in the ore treated is 1·664 per cent., and the cost of treatment per ton of ore is $0·640.

The *Tierras* are treated in very similar furnaces, which, however, differ slightly in the arrangements for their discharge.

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[1] These ores are not weighed, but are estimated by measurement.

### c. Roasting in Reverberatory Furnaces.

At Idria, reverberatory furnaces, constructed by Alberti, are employed for the treatment of some of the smaller and poorer ores, and the results obtained are said to be satisfactory. The ore is introduced into the furnace through a hopper placed at the extremity nearest the chimney, and is divided into three charges, which are gradually worked towards the fire-bridge, while the exhausted matter is withdrawn, through an opening, into an arched chamber beneath. Condensation is effected in cast-iron pipes, through which the volatilized mercury, together with the products of combustion, first pass to a large condensing chamber, and thence back again, through a considerable length of similar pipes, to another chamber, near the furnaces, which is in communication with a high chimney. These tubes are kept cool by a spray of water constantly falling upon them from parallel wooden spouts, pierced with holes, placed above them, and condense rather more than 95 per cent. of the total amount of mercury obtained. Two of these furnaces, placed side by side so as to form one block of masonry, work 13 tons of fine ore and 10 tons of schlich daily, with a loss of about 8 per cent. of the mercury indicated by assay, and with a consumption of 50 cubic feet of wood per cwt. of mercury obtained.

## Decomposition of Mercurial Ores by Lime.

*a.* Gallery of the Palatinate.—In the district of Zweibrücken, Rhenish Bavaria, where considerable quantities of mercury were formerly extracted, a peculiar apparatus called a gallery was employed. The mineral here treated consisted of a mixture of sulphide of mercury and calcite, which was heated in earthen or cast-iron retorts or *cucurbits*, of which several were arranged in one furnace, as shown in fig. 167. The number of cucurbits, A, contained in one gallery varied from thirty to fifty, and to each of these was adapted a stoneware receiver, B, partially filled with water. Into each of the retorts were introduced from 56 to 70 lbs. of cinnabar and from 15 to 18 lbs. of quicklime, a mixture which should fill about two-thirds of its capacity.

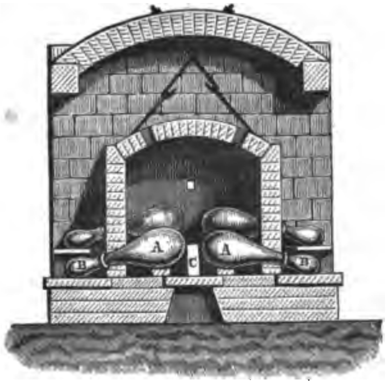

Fig. 167.—Gallery; transverse section.

The sulphide of mercury was in this case decomposed by the lime, sulphide of calcium and sulphate of calcium being formed, and the liberated metal was condensed in the stoneware bottles. The

fuel employed was pit-coal, burnt on a small grate situated at C. The dome was perforated with openings for the purpose of creating a draught.

*b.* Iron Retorts.—With the view of obviating the inconvenience and loss experienced by the older methods of distilling mercury, an apparatus was erected in 1847, under the direction of the late Dr. Ure, at Landsberg, in Rhenish Bavaria. This arrangement consisted of a series of retorts, which were set in masonry, precisely in the same way as those used in the manufacture of coal gas, and were fitted at one end with an eduction tube, and at the other with an air-tight stopper, kept in its place by an iron screw.

Both in Bavaria and at Ripa, in Tuscany, where this apparatus was also introduced, it has long since been superseded, and at the present time it appears to be nowhere in use. Retorts on a somewhat similar plan have been tried in some of the mercury mines of California, but were not found economical; at the Enriquetta Mine, rotating retorts were formerly employed.

## BISMUTH.

Bismuth possesses a greyish-white colour, but at the same time presents a distinctly red tint when compared with zinc, antimony, or any of the whiter metals. It is brittle, and consequently cannot be drawn out under the hammer, and when broken presents a highly crystalline fracture. Very beautiful crystals of this metal are obtained by fusing a considerable quantity in an earthen crucible, and afterwards setting it aside and allowing it to cool gradually. For this purpose the crucible in which the fusion has been effected should be removed from the fire to a sand-bath, and covered with a hot-iron plate, on which are placed a few pieces of ignited charcoal. At the expiration of a certain time, the external crust of solidified metal is pierced by a hot iron, and the interior portions, which still retain the liquid form, are rapidly poured out. The upper crust is now removed, and crystals of bismuth are found coating the sides of the vessel. These are really rhombohedra, but having angles of nearly 90°, they have the appearance of cubes, and from a slight covering of oxide, varying in its thickness, they frequently assume beautiful prismatic colours.

Commercial bismuth is never pure; but as the other metals with which it is associated are commonly more oxidizable than itself, they may, in a great degree, be separated from it by fusing the powdered metal in an earthen crucible, with one-tenth of its weight of nitrate of potassium. On heating this mixture until the nitre has been completely decomposed, a portion of the bismuth, together with the major part of the impurities, will have been oxidized and will remain with the slag, while a button of purified bismuth collects in the bottom of the crucible.

To more completely purify bismuth, dissolve the crude metal in nitric acid and concentrate by evaporation. Pour the clear concentrated solution into a large quantity of distilled water, and wash the basic nitrate which is precipitated by decantation. Boil with a very weak solution of caustic potash to remove traces of arsenic, &c., wash and dry. Mix the dried basic nitrate with its own weight of black flux, and fuse it at a moderate heat in an earthen crucible. On breaking the crucible, after cooling, a button of nearly pure bismuth will be found at the bottom.

This metal fuses at a temperature of 258° C. It is volatile at a high heat, and may be distilled. Bismuth is placed by Faraday at the head of diamagnetic substances; its thermal and electric conductivity are lower than those of most other metals. At a white heat bismuth boils, and is sublimed, and at this temperature is stated to decompose the vapour of water; it is not affected by exposure to dry air, but when placed in a humid atmosphere gradually becomes covered with a thin pellicle of oxide. When strongly heated in air, bismuth burns with a bluish flame and gives off fumes of a light yellow colour.

It is attacked with difficulty by concentrated hydrochloric acid. Sulphuric acid, unless concentrated and hot, does not attack it, and in this case sulphurous anhydride is evolved. Nitric acid attacks it with great facility, with the formation of a soluble nitrate of bismuth.

## BISMUTH ORES.

Bismuth occurs native, and also in combination with the closely allied element tellurium, as well as with silver, gold, sulphur, and oxygen. Its ores readily fuse before the blowpipe, and in the oxidizing flame afford an oxide by which the charcoal support is stained of a brownish-yellow colour.

**NATIVE BISMUTH**; *Bismuth natif; Gediegen Wismuth.* Hexagonal, rhombohedral.—Is found massive, granular, reticulated, or arborescent. Colour greyish-white, inclining to red; lustre metallic, and streak unchanged. Frequently contains small quantities of arsenic, and is often associated with silver, and sometimes with iron.

Native bismuth accompanies various ores of silver, lead, zinc, cobalt, and nickel, and frequently occurs in veins traversing either gneiss or clay-slate. Its principal localities are in Australia; the silver and cobalt mines of Saxony and Bohemia, at Altenberg, Schneeberg, Annaberg, Joachimsthal, and Johanngeorgenstadt; at Lölling in Carinthia, and at Fahlun in Sweden. Native bismuth occurs in small quantities at Huel Sparnon, near Redruth in Cornwall; at Carrick Fell in Cumberland; at Alva, in Stirlingshire, Scotland; and in Bolivia. Native bismuth supplies nearly the whole of this metal which is employed in the arts; the greater portion was formerly derived from the mines of Schneeberg, where it was found associated with ores of cobalt.

Bismuth is also found in combination with other bodies, but these compounds are by no means of common occurrence.

*Sulphide of Bismuth* occurs in Cumberland, Cornwall, Saxony, Sweden, and in South Australia. It is found both in the massive state and in the form of acicular crystals. Its composition is $Bi_2S_3$, or bismuth 81·3, sulphur 18·7. This mineral is by no means plentiful, although its localities are comparatively numerous.

*Silicate of Bismuth* occurs in minute dodecahedral crystals of a dark hair-brown or wax-yellow colour. Its more general appearance is that of implanted globules which rarely exceed the size of a pin's head. A specimen of this mineral, from Schneeberg in Saxony, was found to be composed of oxide of bismuth, 58·8; silica, 23·8; arsenic anhydride, 2·2; gangue, 9·1; cobalt, copper, and iron, 5·9.

*Needle Ore* or *Patrinite* is a sulpho-bismuthide of copper and lead sulphides, which is found in the mine of Klutscheffsky, near Beresof, in Siberia, in acicular crystals of a yellowish-white colour, and contains from 28 to 37 per cent. of bismuth.

*Tetradymite* is a compound of bismuth and tellurium, and occurs in association with other tellurium ores and gold in Merionethshire, Hungary, Georgia, Colorado, and California.

*Oxide of Bismuth* occurs as a pulverulent coating on some of the other ores of this metal; it is found in Bohemia, in Siberia, at St. Agnes in Cornwall, &c. It is of a yellowish-green colour, and contains 86 per cent. of bismuth. A vein, containing ores of bismuth and wolfram, was formerly worked near Meymac, Dep. of Corrèze, France. The bismuth occurs as oxide, associated with native metal and sulphide.

*Carbonate of Bismuth* occurs at St. Agnes, at Schneeberg, and at Johanngeorgenstadt.

About 40 tons of bismuth ores, containing 60 per cent. of metallic bismuth, are annually imported into this country, chiefly from Australia.

## ASSAY OF BISMUTH ORES.

Assays of the ores of bismuth are conducted like those of the oxidized ores of lead. When the substance operated on contains metallic bismuth only, no reducing flux is, theoretically, required; but as there is, in almost all cases, a portion of oxide present, a little powdered charcoal should be added. On account of the volatility of this metal, it is of importance that a readily fusible slag should be obtained, and for this purpose large quantities either of sodium carbonate, borax with charcoal, or of borax with black flux, should be employed.

In the wet way, bismuth is usually precipitated from its solutions by carbonate of ammonium, which when added in excess throws down the whole as carbonate, provided the liquid be allowed to stand for several hours in a warm place. The precipitate, after being washed and dried, is separated from the filter and ignited in a porcelain crucible; the filter is burnt separately, and the residue added to the ignited precipitate. This consists of the oxide $Bi_2O_3$ containing 89·74 per cent. of metal.

When sulphuric or hydrochloric acid is present in the solution, carbonate of ammonium must not be employed for precipitation, since the precipitate would, in the former case, contain basic sulphate, and in the latter, oxychloride of bismuth. In such cases bismuth must first be precipitated by sulphuretted hydrogen, and the resulting sulphide attacked by nitric acid. From the solution thus obtained bismuth may be precipitated by carbonate of ammonium.

## METALLURGY OF BISMUTH.

Schneeberg Process.—The bismuth of commerce is chiefly obtained from the native metal, of which a large proportion was formerly procured from the mines of Schneeberg, in Saxony. The metallurgical treatment of these ores is extremely simple, as it is sufficient to heat them in closed vessels; by which treatment the metal becomes fused and flows out into proper receivers, while the gangue and infusible impurities remain behind.

The Schneeberg process of liquation is effected in cast-iron retorts, *a*, *b*, fig. 168, set in an inclined position in brickwork, A, and provided with a grate, *g*, for the fuel employed. The ore treated is sorted by hand, broken into pieces of the size of a hazel nut, and separated as much as possible from associated gangue. The charge of each tube consists of about 56 lbs. of broken ore, which is introduced at *a*, and occupies three-fourths of its length and rather more than one-half its diameter. The iron door at the end, *a*, is now shut, and, when the whole of the tubes in the series have been charged in the same way, heat is applied. The liquid metal soon begins to flow through the apertures, *b*, left in the lower ends of the tubes, and falls into small pots, *c*, kept slightly heated by a few pieces of ignited charcoal, introduced into a space left beneath them for that purpose. Whenever the metal ceases to run freely, an iron rod is inserted through the aperture, *b*, and the ore is moved about in the retort in order to remove any obstruction. The

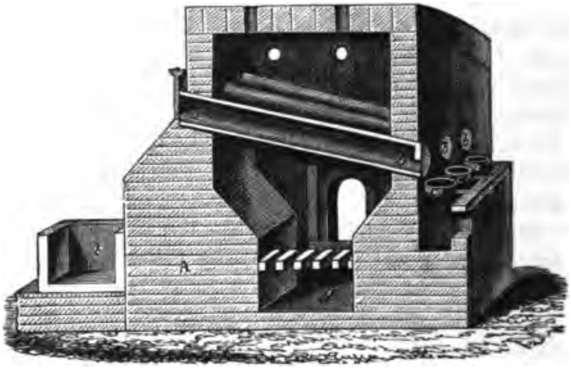

Fig. 168.—Bismuth Liquation-Furnace; vertical section.

fuel employed is wood, and each operation requires about an hour for its completion.

As soon as the flow of metal has entirely ceased, the residuum is scooped out with iron rakes into the water-trough, *t*, and a fresh charge of ore at once introduced into the retorts. The contents of the pots, *c*, are dipped out with ladles, and cast into ingots varying from 25 to 50 lbs. in weight.

By this apparatus 20 cwts. of ore may be treated in eight hours, with a consumption of 63 cubic feet of wood. The annual production of bismuth at Schneeberg formerly amounted to about 5 tons, but it is now considerably less.

JOACHIMSTHAL PROCESS.—At Joachimsthal, ores containing from 10 to 30 per cent. of bismuth are treated in large clay crucibles. The ore is ground and mixed with 28 per cent. of iron-turnings or other finely divided scrap-iron, 15 to 20 per cent. of carbonate of sodium, 5 per cent. of lime, and 5 per cent. of fluor-spar. The crucibles employed are 23 inches in height and 16 in diameter at the mouth; they are filled with the mixture above specified, and its fusion is effected in a wind-furnace. When in a state of tranquil fusion the contents of the pots are poured into inverted conical moulds, in the bottom of which the bismuth collects; this is covered by a speiss, which, in addition to cobalt and nickel, contains about 2 per cent. of bismuth. The speiss is separated from the bismuth for subsequent treatment, and the slag, which covers both, is thrown away. The bismuth thus obtained contains a certain amount of silver, which may be separated by subjecting the mixture to cupellation, and subsequently reducing the oxide of bismuth produced. Alloys of silver and bismuth work on the cupel quite as well as those of silver and lead.

PRODUCTION OF BISMUTH AT FREIBERG.—I am indebted for the following description of a method employed at Freiberg in 1870, for the extraction of bismuth from argentiferous ores, to Mr. W. M. Hutchings of Chester, who, when a pupil at the Royal Mining Academy, possessed facilities for making himself acquainted with the process.

None of the Freiberg ores contain an appreciable amount of bismuth, and indeed the quantity present in any of them is so small that it was never detected by direct analysis. On an analysis being made, however, of the hearth-bottom of the silver refinery on which a charge of *Blicksilber* from the German cupelling-furnace had been refined, it was found to be rich in bismuth. Further analyses proved this to be always the case; some of the hearths containing as much as 20 to 25 per cent. of bismuth, showing that the small quantity originally present in the ores had become concentrated in the Blicksilber, and had finally passed into the refining-hearth. A process for its extraction was consequently introduced, and was carried on at intervals, whenever a sufficient accumulation of material had been collected. For this purpose the hearth was finely ground and passed through a sieve, to remove all metallic shot, The fine powder was then submitted to the extraction process about to be described.

At a later period a class of ores from another part of the Erzgebirge, containing more bismuth, was sent to be smelted at the Freiberg works; these ores contained nickel and cobalt, which were concentrated in the speiss produced in smelting. In order to obtain the bismuth from these ores it was no longer sufficient to treat the refining-hearth only by the extraction process, since towards the close of a cupellation the litharge produced was so rich in bismuth that it was found necessary to keep it apart and to send it to the bismuth works. It was also found advantageous to stop the operation of cupellation when the charge was enriched up to 80 per cent. of silver, and to transfer the alloy to the refining-furnace, in which cupellation is completed and the silver refined; both the litharge and the hearth contain from 5 to 20 per cent. of bismuth. This litharge, as well as the hearth, is finely ground, and both are treated as follows:—

The powdered hearth or litharge is treated with hydrochloric acid in salt-glazed earthenware pots, 2½ feet high, having an internal diameter of 1 foot 11 inches at bottom and 2 feet 8 inches at top. Twelve of these pots are ranged upon a platform at one end of the room, with a water-pipe placed above and parallel with them, so that water can be let into each as required. A steam-pipe is also provided, in order that, when necessary, steam may be blown into the several pots to heat their contents.

From 80 to 100 lbs. of hearth, or litharge, are placed in each pot, and measured quantities of acid and water introduced, in accordance with the previously determined richness in bismuth of the material operated on. Thus, for example, 100 lbs. of hearth may receive 120 lbs. of acid and 45 lbs. of water. The acid used is common yellow hydrochloric. The reaction is strong, and a considerable amount of heat is evolved; the mass is actively stirred and is prevented from forming lumps. Stirring is frequently repeated during seven or eight hours, after which more acid is added, in quantity regulated by the amount of bismuth present, so that the pot can be filled with water without any precipitation of bismuth oxychloride taking place.

The vessel, when so filled, is allowed to stand thirty-six hours, to let the contents thoroughly settle. The clear liquid is then removed by leaden syphons, and conveyed along wooden gutters to the precipitating tubs, which stand upon a platform on a lower level. These tubs, made of pine-wood, are 4 feet 8 inches high, with an internal diameter of 3 feet 7 inches, and sides 2½ inches in thickness. Each tub has two taps, one quite at the bottom and the other some 5 inches higher up. A wooden gutter runs over all the tubs, and has a hole and plug corresponding with each. The liquid syphoned out of the extraction-pots is conveyed to this main channel by short movable gutters, and can, by plugs, be let into any one of the tubs in the series. At the same time that this liquid is run in, water is introduced, from a pipe extending over the tubes, in sufficient quantity to ensure the precipitation of the bismuth as oxychloride; the whole is stirred and the precipitate allowed to settle.

The residues in the extraction-pots are again treated with acid and water, and the solution precipitated as before; this treatment is repeated until the solution obtained is so weak that on adding a large quantity of water no appreciable precipitate is formed. The residues are then removed, drained, dried, and when a sufficient quantity has accumulated, are passed through the blast-furnace.

When the oxychloride of bismuth has completely settled in the precipitating tubs, the clear liquid is let off by the upper tap, and conveyed along gutters into large wooden settling tanks, in order to catch any small quantity of the precipitate which may be carried over. From these the liquors are run off, leaving the precipitate in the bottom. Another charge of liquid is now run into the precipitating-tub and treated in the same way; several precipitations thus take place in each tub, until the precipitate nearly fills the space between the taps. After the clear liquid has been run off by the upper tap, the precipitate is stirred, drawn off by the lower one, and conveyed on to suitable filters. Each filter is connected with a tub, into which the filtered liquid runs before going to the settling tanks.

This first precipitate obtained upon the filters is not sufficiently pure to be reduced at once to the metallic state. It is, therefore, taken from them, treated with acid and water, and re-precipitated; for this purpose there is a special set of three smaller pots and three separate precipitating tubs.

The second precipitate is almost pure; this is dried, and afterwards reduced by fusion in iron crucibles, heated in a wind-furnace with 50 per cent. of carbonate of sodium, 7 per cent. of charcoal-powder, and 3 per cent. of powdered glass.

The bismuth thus obtained is refined by re-melting in iron crucibles, and the removal of the scum which forms. The resulting bismuth is of good commercial quality.

The bismuth ores treated in the United Kingdom contain on an average 60 per cent. of metallic bismuth, and are generally fused in black-lead pots, with the addition of borax, sodium, carbonate, and a little crude tartar.

Bismuth unites readily with tin and lead, forming a series of alloys known as *fusible metals*, whose melting-points are considerably lower than those of their constituents. This curious property was first noticed by Sir Isaac Newton, who described an alloy melting near the boiling-point of water; and subsequent observers have shown that by varying the proportions of the constituents, and especially by a small addition of cadmium, the fusibility may be greatly increased. The complete series of these alloys is as follows:—

| Name. | Bi. | Pb. | Sn. | Cd. | Melting-Point. | Remarks. |
|---|---|---|---|---|---|---|
| | | | | | C. | |
| 1. Newton's . | 50 | 31·25 | 18·75 | ... | 94°·5 | ... |
| 2. Rose's . . | 50 | 28·10 | 24·64 | ... | 100° | $Bi_2Sn_2Pb$ |
| 3. Darcet's . | 50 | 25·00 | 25·00 | ... | 93° | ... |
| 4. ,, . | 50 | 20·00 | 30·00 | ... | 91°·6 | ... |
| 5. Wood's . . | 50 | 14·00 | 14·00 | 12·00 | 66–71° | ... |
| 6. Lipowitz's . | 50 | 27·00 | 13·00 | 10·00 | 60° | Softens at 55° |

Fusible metal, like pure bismuth, expands in solidifying, and is used for taking impressions or *clichés* of wood-blocks and other engraved surfaces for reproduction by the electrotype process, as well as in safety-plugs for steam-boilers, which melt if the water becomes overheated through excessive pressure. A small addition of bismuth to pewter renders it hard and sonorous. The salts of bismuth are somewhat extensively used in medicine.

## LEAD.

Lead is a soft metal of a bluish-grey colour, and when recently cut possesses a strong metallic lustre; on exposure to the air it becomes rapidly tarnished, and acquires a superficial coating of plumbous carbonate.

Lead is both malleable and ductile, possessing the former property to a considerable degree; but its tenacity is inferior to that of nearly all the other ductile metals. It is flexible and inelastic, and fuses at about 325° C. When slowly cooled, imperfect octahedral crystals are readily obtained. At a red heat, lead becomes sensibly volatile, but not to a sufficient extent to admit of its distillation.

When kept in a state of fusion, in contact with the air, rapid oxidation takes place. At first the surface of the metallic bath becomes covered by an iridescent pellicle, which is quickly converted into a powder of a reddish-yellow colour. At a red heat this oxidation of the metal proceeds with great rapidity; and it becomes necessary, in order to continue the operation, that the oxide, which gradually melts, should be removed for the purpose of exposing a fresh metallic surface.

Lead, exposed to the influence of a damp atmosphere, quickly absorbs oxygen, and when acid vapours are present this action is much accelerated. Oxidation is induced by the presence of carbonic anhydride, which gives rise to the formation of a white carbonate of lead. Distilled water determines the oxidation of the metal, and from this cause leaden cisterns are rapidly corroded when used as reservoirs for soft water. A bar of lead, placed in distilled water and exposed to the air, becomes rapidly covered with a white coating of hydrated oxide, which is subsequently converted into a hydrated carbonate of lead, frequently forming distinct nacreous scales on the surface of the metal. In such cases the water is invariably found to hold a portion of lead in solution, which is readily shown by its becoming brown on passing through it a current of sulphuretted hydrogen.

From the tendency exhibited by lead to form soluble salts, it ought not to be used for the manufacture of tanks in which water for domestic purposes is to be kept, since, from the poisonous nature of these compounds, disastrous effects not unfrequently result.

The action of water on lead is, however, much diminished by the presence of small quantities of various salts, and particularly calcium sulphate, which has the property of preventing, to a great extent, the oxidation and solution of this metal.

The lead of commerce often approaches a state of chemical purity, and is then extremely soft and malleable. When lead of still greater purity is required, it may be procured by reducing, in a carbon-lined crucible, oxide of lead, obtained by the calcination of crystallized nitrate of lead. Lead is somewhat feebly attacked by hydrochloric acid, even when concentrated and boiling. Weak sulphuric acid does not act on lead when air is excluded; but if heated in very strong sulphuric acid, $SO_2$ is evolved and lead sulphate is slowly formed. The proper solvent for lead is nitric acid, which forms with it a salt readily crystallizing, on cooling, in opaque octahedra.

## LEAD ORES.

Lead is very rarely found in a native state, but usually in combination with one of the non-metallic elements, particularly with sulphur. It also occurs in combination with oxygen, selenium, arsenic, tellurium, and with various acids. The ores of lead are fusible before the blowpipe, and when fluxed with a little carbonate of sodium on a charcoal support yield a globule of metallic lead. The metal thus obtained gives off fumes, particularly when heated in the outer flame, and stains the charcoal of a yellow colour.

NATIVE LEAD; *Plomb natif; Gediegen Blei.* Cubic.—The characters of native lead are precisely similar to those of ordinary commercial lead. It is a very rare substance, of which specimens are said to have been found, associated with galena, in County Kerry, Ireland, and in an argillaceous rock near Cartagena, Spain. Native lead is also stated to have been met with at Alston Moor, in Cumberland, where it is supposed to occur, with galena, in a siliceous rock.

OXIDE OF LEAD; *Massicot; Bleiglätte.* Rhombic.—Is a pulverulent mineral of a bright red colour, sometimes mixed with yellow, and is a mixture of different oxides of lead, affording a metallic globule when heated on charcoal before the blowpipe. It is sometimes a volcanic product, but occurs associated with galena, and is found in small quantities in various lead mines. From the comparative rarity of this ore it is of no importance to the metallurgist.

**Chloride of Lead**; *Plomb chloruré; Salzsaures Blei.* Rhombic.—This rare mineral, known as *Cotunnite*, $PbCl_2$, is found among the lavas of Vesuvius. An oxychloride of lead, termed *Mendipite*, occurs in the Mendip Hills, in the form of lamellar, shining masses, of a greyish-white colour, deposited on a matrix of black oxide of manganese. It has a specific gravity of 7·07. When treated before the blowpipe it decrepitates, and fuses into a globule of a yellowish-white colour; if heated on charcoal, metallic lead is obtained. Another oxychloride of lead, and a chloro-carbonate of lead, occur in Derbyshire.

**Sulphide of Lead**; *Galena; Galène; Bleiglanz.* Cubic.—This mineral occurs principally in cubes, which are often of considerable size; its cubic cleavage is extremely perfect. It more rarely occurs in a finely granular state, and is sometimes found in fibrous masses. Compact specimens, although occasionally met with, are of comparatively rare occurrence. Its colour and streak are lead-grey; fragile; lustre metallic; specific gravity from 7·3 to 7·7. When pure, it contains lead 86·55, and sulphur 13·45 per cent., corresponding to the formula, PbS. The lead in this mineral is invariably associated, to a greater or less extent, with silver. When silver is present in considerable quantity the ore receives the name of argentiferous galena, and becomes a valuable source of that metal.

The analysis of an argentiferous galena from Schemnitz afforded Beudant the following results:—

| | |
|---|---|
| Pb | 79·60 |
| Ag | 7·00 |
| S | 13·40 |
| | 100·00 |

or about 2,300 ounces of silver per ton of ore. This is rarely realized for any large quantity of ore; the average yield of lead and silver of galena from different districts, as prepared for the smelter, being as follows:—

| | Lead, per cent. | Silver, ounces per ton. |
|---|---|---|
| North of England | 70–77 | 8 |
| Linares, Spain | 70 | 10 |
| Harz | 62 | 30 |
| Freiberg | 55 | 45 |
| Isle of Man | 75 | 20–60 |
| Cornwall | 70 | 50 |
| Przibram | 49 | 100 |

In addition to sulphide of silver, galena sometimes contains variable quantities of antimony. This substance appears to alter in a certain degree the character of the mineral; those specimens of which the laminæ are curved, as well as those which present a bright steely fracture, often contain antimony.

Galena occurs in granite, limestone, in argillaceous, and in sandstone rocks, and is frequently associated with ores of copper and zinc. The matrix on which this ore has been deposited is, in the majority of cases, either quartz, calcite, fluor-spar, or barium sulphate. The lead mines of the West of England occur in clay-slate; those of Derbyshire and the

North of England are principally in limestone, as are also the extensive deposits of Bleiberg and of the neighbouring districts. In the Upper Harz, and at Przibram, in Bohemia, the lead mines are in clay-slate; at Freiberg, in Saxony, in gneiss; at Sala, in Sweden, in crystalline limestone; and at Leadhills, in Scotland, in the older grits. Valuable deposits of galena are worked in various parts of France, and particularly at Huelgoët and Poullaouen, in Brittany; at Pontgibaud, Puy-de-Dôme; and at Vialas, in the department of Lozère. In Spain, sulphide of lead is found in Catalonia and Granada, in the granite hills of Linares, province of Jaen, and elsewhere. Galena occurs in Belgium; in Savoy; in Bohemia, at Joachimsthal, where the ore is principally worked for silver; and in Siberia, where argentiferous galena occurs in limestone in the Daouria Mountains. Extensive deposits of this ore are likewise found in United States of America.

*Cuproplumbite* is a variety of galena containing 24·5 per cent. of sulphide of copper. It is a rare mineral, obtained from Chili.

*Dufrenoysite* is a sulpharsenide of lead of a dark steel-grey colour, from the dolomite of St. Gothard.

*Selenide of Lead* or *Clausthalite*, is a mineral of a lead-grey colour and granular fracture. When heated before the blowpipe it gives off the odour of horse-radish. It occurs in quantities too small to render it of practical value as an ore of lead.

CERUSSITE; *Carbonate of Lead; Plomb carbonaté; Weissbleierz.* Rhombic.—This mineral generally possesses a white colour and an adamantine lustre. It is found in crystals analogous to those of aragonite, in radial and compact masses, in concretions, and in amorphous deposits. All these varieties, with the exception of that last-mentioned, possess the peculiar lustre belonging to white lead. It sometimes happens that crystallized specimens of this substance are nearly black; this arises from the presence of small quantities of sulphide, probably due to the action of sulphuretted hydrogen, resulting from the decomposition of galena, with which carbonate of lead is generally found associated. It is an extremely brittle mineral, and when amorphous exhibits a conchoidal fracture. When treated with nitric acid it dissolves with evolution of $CO_2$; before the blowpipe it decrepitates, but when heated on a charcoal support affords a button of metallic lead. Its specific gravity varies from 6·46 to 6·58.

Two specimens of this mineral afforded on analysis the following percentage results:—

| | Crystals from Leadhills. By Klaproth. | Crystals from Teesdale. By J. A. Phillips. |
|---|---|---|
| PbO . . . . . | 82 | 83·55 |
| $CO_2$ . . . . . | 16 | 16·52 |
| | 98 | 100·07 |

This is a carbonate of lead, having the formula $PbCO_3$ or $PbO.CO_2$. The amorphous and friable varieties are generally more or less contaminated with siliceous and earthy impurities. Cerussite is found in splendid crystals at Leadhills, at Wanlock Head, in Derbyshire, and in some of the Cornish mines, as well as in many other localities. When abundant it forms a valuable ore of lead, sometimes yielding above 75 per cent. of that metal. From its dissimilarity to the other ores of lead, it was for a long time considered by miners to be of no value; large quantities which had been formerly buried in rubbish were subsequently excavated and worked with great advantage in many of the Spanish mines, as also at different points in the valley of the Mississippi, and at Leadville, Colorado, United States of America.

ANGLESITE; *Sulphate of Lead; Plomb sulfaté; Bleivitriol.* Rhombic. —The crystalline forms are closely allied to those of heavy spar, and are often slender and implanted. Specimens of sulphate of lead in amorphous masses and in lamellar and granular fragments are also found. It is colourless, sometimes inclining to grey or green. Lustre adamantine, vitreous, or resinous. May be either opaque or perfectly transparent. When pure, it consists of 73 per cent. of oxide of lead, and 27 of sulphuric anhydride. If heated with sodium carbonate before the blowpipe it affords a globule of metallic lead. Its composition is represented by the formula $PbSO_4$, or $PbO.SO_3$. This mineral is generally associated with galena, by the oxidation of which it appears to be formed.

Fine specimens of this ore are found at Parys Mountain, Leadhills, and Wanlock Head, as well as at Huelgoët in France, in the Island of Sardinia, and in the States of Missouri and Wisconsin in America; but it does not often occur in sufficient quantities to be regarded as an important ore of lead. Its density is about 6·3.

*Linarite* is a blue hydrated double sulphate of lead and copper, sparingly found at Leadhills and at Roughton Gill.

PYROMORPHITE; *Phosphate of Lead; Plomb phosphaté; Buntbleierz.* Hexagonal.—This mineral occurs in hexagonal prisms, of a bright green or brown colour, which have a lateral cleavage, and are often nearly transparent. It has a specific gravity varying from 6·5 to 7·1, and affords a white streak.

Besides being found in crystals, it sometimes occurs in mammillary and reniform masses, with a radiated structure. In its purest state this mineral is composed of PbO 74·1, $P_2O_5$ 15·7, Pb 7·6, and Cl. 2·6 per cent.

This composition indicates the proportion of three molecules of phosphate of lead to one of chloride of lead; formula $3(3PbO.P_2O_5).PbCl_2$, or $Pb_5(PO_4)_3Cl$. Phosphate of lead is found in many of the lead mines in this country; and particularly in those of Cornwall, Leadhills, and Wanlock Head. The phosphate of lead from Huelgoët, in Brittany, contains alumina.

*Mimetisite*, *Kampylite*, and *Hedyphane* are arsenates of lead, much resembling in appearance the phosphate of that metal, but when heated evolving the odour of garlic. They are analogous in crystalline form and chemical constitution to Pyromorphite $P_2O_5$, being replaced by $As_2O_5$, and PbO to some extent by CaO.

*Chromate of Lead* is a mineral of a bright red colour, which crystallizes in rhombic prisms, and blackens before the blowpipe. When heated on charcoal it forms a shining slag containing numerous globules of metallic lead. It has a specific gravity of about 6·0.

A specimen of chromate of lead, analyzed by Berzelius, gave the following results:—

| | |
|---|---|
| PbO | 68·50 |
| $CrO_3$ | 31·50 |

This mineral is a simple chromate expressed by the formula $PbCrO_4$, or $PbO.CrO_3$. Chromate of lead is the "chrome-yellow" of painters, but is for this purpose artificially prepared by adding a solution of chromate of potassium to a soluble salt of lead. Native chromate of lead occurs in small quantities only, and is chiefly obtained from Brazil and from Beresof in Siberia. *Melanochroite* is another chromate of lead, and *Vauquelinite* is a chromate of lead and copper.

*Wulfenite* or Molybdate of Lead is a very brilliant yellow mineral, crystallizing in tetragonal forms. It is of the composition $PbO.MO_3$, with PbO 61·4 and $MO_3$ 38·6 per cent. It occurs most abundantly at Bleiberg, Carinthia, and in Utah and Nevada, and is the principal source of molybdic acid.

*Plumbo-resinite* is a rare ore of lead, obtained at Huelgoët, in Brittany, and from the Missouri mines in the United States of America. A specimen of this substance from Huelgoët, analyzed by Berzelius, was found to be constituted as follows:—Oxide of lead, 40·14; alumina, 37·00; water, 18·80; insoluble gangue, 2·60. This mineral has a yellowish or reddish-brown colour, and possesses a lustre much resembling that of gum arabic.

The other minerals containing lead are very rare, and in no instance occur in sufficient abundance to allow of being treated as ores of this metal.

## DISTRIBUTION OF LEAD ORES.

The ores of lead are abundantly distributed through the geological series, but appear to be most abundant in rocks of Silurian and Carboniferous age, and frequently occur in deposits which cannot be regarded as true veins. Lead veins are often rich in one stratum of rock, and become suddenly impoverished on entering another differing from it in composition. Galena always contains a certain amount of silver, but lead ores are not, generally, argentiferous to any considerable extent, unless they occur in crystalline or metamorphic rocks. The more argentiferous ores are, for the most part, found in true veins occurring in the older rocks, and these, although not so productive for lead as deposits in limestone, are generally more persistent in depth.

The ores of lead found near the surface embrace various oxidized combinations, resulting from the decomposition of galena; among these carbonate, sulphate, and phosphate of lead are the most common.

The lead-producing districts of the United Kingdom are scattered over England, Wales, Scotland, and Ireland; but that of the north of England is, from the quantity of ore raised, the most important. It lies chiefly in the vicinity of Alston Moor, where the three counties of Northumberland, Durham, and Cumberland meet, and where the ore is obtained from veins enclosed in Carboniferous Limestone. A nearly horizontal bed of eruptive rock, known to the miners as the "Whin Sill," is intercalated between the limestone in an irregular manner. The principal workings are on *rake-veins*, or true lodes; but there are other classes of deposit, known respectively as *pipe-veins*, *flat-veins*, &c.

Rake-veins commonly exhibit the usual characteristics of regular veins, although they do not always descend through the strata in an uninterrupted course, but are arranged in zigzags, one portion having a general parallelism with other parts of the same vein above or below it, and being connected with it by horizontal deposits.

The lead-region of Derbyshire is in many respects similar to that above described, but is more complicated in its details, being much broken up by faults; and instead of one bed of eruptive rock, as in Cumberland, there are three. The ordinary gangue of these veins is calcite, fluor-spar, and barium sulphate.

The lead mines of Cornwall and Devon are worked on true veins, and the ores raised contain a notable amount of silver, but the production has of late years considerably fallen off. At the time Borlase wrote (1758) only one lead mine was worked in Cornwall, and in 1839 the whole produce of the county was somewhat below 180 tons. In the years from 1845 to 1850 over 10,000 tons were annually raised, from 3,000 to 4,000 tons of metallic lead being produced annually from East Huel Rose alone. This mine has long since ceased to be productive, and according to 'Official statistics, the present annual production of Cornwall is about 350 tons of lead, containing 2,500 ounces of silver. In Devonshire the Combe Martin and Beer Alston Mines, which formerly yielded ores containing from 80 to 140 ounces of silver per ton, have long since ceased to be extensively worked. The Snailbeach Mines in the county of Salop, the Grassington Mines in Yorkshire, and the Minera Mines in Denbighshire, have been celebrated for their large production.

The lead-region of Cardiganshire and Montgomeryshire extends over a length of about forty miles, and varies from five to twenty-two miles in width. The usual strike of the lodes, which are enclosed in rocks of Silurian age, is east-north-east; the gangue consists chiefly of fragments of slate cemented together by quartz and calcite.

The total quantities of lead ore raised and sold in the United Kingdom during 1885, and of lead and silver produced therefrom, were as follow:—

| | |
|---|---|
| Lead ore | 51,302 tons. |
| Lead | 37,687 „ |
| Silver | 320,520 ounces. |

In Belgium, galena and other ores of lead are found, associated with zinc, in limestone. The annual production of the country is probably equivalent to about 2,500 tons of lead.

In Germany, lead mines of some importance are worked in the Siegen district. The veins of the Upper Harz are concentrated in two principal groups, one near Clausthal, and another in the vicinity of St. Andreasberg. In the neighbourhood of Clausthal and Zellerfeld the veinstone is chiefly made up of a breccia of country rock cemented together by calcite, carbonate of iron, quartz, and heavy spar. The most abundant ore is argentiferous galena, with small quantities of copper ore and blende; in some cases the ores widen out into a stockwork 300 feet in width, and from such aggregations of narrow veins, rich returns of ore are not unfrequently obtained. The system of veins in the neighbourhood of St. Andreasberg is included within a space about a mile in length and two-thirds of a mile in width; in addition to argentiferous galena, they yield silver ores proper, including pyrargyrite and light-red silver ore.

Besides the mines of St. Andreasberg and Clausthal, there are in the Harz those of Rammelsberg, situated in the neighbourhood of Goslar. Here the principal bed of ore dips in the direction of the enclosing slates, and, at a depth of about 40 feet, sends off a branch at a considerably less angle. The length of this deposit is about 1,800 feet, and its greatest thickness 150 feet, but these dimensions decrease in depth. This remarkable deposit is almost entirely composed of sulphides of iron, zinc, lead, and copper, without any notable admixture of gangue. There are also extensive workings in the great lead-bearing sandstones of Commern and elsewhere, in Rhenish Prussia, where from rock yielding only 1½ to 2 per cent. nearly 55,000 tons of lead are annually produced.

In Nassau lead ores are raised from a group of veins extending from Holzappel on the Lahn to Welmich and Werlau on the Rhine; about thirty small mines are worked on these veins, and are estimated to yield 800 tons of lead annually.

The total production of lead ore in Germany was, in 1881, about 164,000 tons.

A considerable portion of the lead produced in the Austrian Empire is obtained from the mines of Bleiberg and Raibl in Carinthia. The village of Bleiberg is situated near Villach, in the Carinthian Alps, and mines extend from Bleiberg along the valley of the Nötsch to a distance of five miles. The ore, which is galena and carbonate of lead, with blende and calamine, occurs in deposits in a rock believed to be of the age of the Muschelkalk.

At the important mines of Przibram in Bohemia the galena forms contact-deposits on the sides of dioritic veins occurring in rocks of Lower Silurian age. The metalliferous portions of these veins, which are generally too poor to repay the expense of working until a depth of fifty fathoms has been attained, consist of a mixture of galena, sulphide of antimony, blende, and iron pyrites, with a little grey copper ore. The total average production of the Austrian Empire is estimated at about 9,000 tons of metallic lead annually.

The annual production of lead in Russia, Sweden, and Norway is comparatively small.

Spain has long been celebrated for her lead mines, which have been described by Strabo, Diodorus, and Pliny, as exceedingly numerous and extensive. Under the Moorish dominion mining operations were conducted with considerable activity, but upon their expulsion from the country the art appears to have rapidly fallen into decay.

The discovery of America and of its mineral riches, which took place shortly after the departure of the Moors, caused the mines of Spain to be comparatively neglected; but after the loss of her American colonies it was found necessary to make an effort for the development of her own mines. By a decree of Ferdinand VII. the mines of Spain were, in 1825, laid open on tolerably liberal conditions to the enterprise of all, whether natives or foreigners, and in 1849 this law was supplanted by a new one, by which still further privileges were conferred.

One of the first points to which Spanish mining enterprise was directed, after the promulgation of the ordinance of 1825, was the lead-district of the Sierra de Gador, in the province of Almeria, where, in 1826, operations had been commenced on above three thousand different grants. For a time the production of this region was very large; in the year 1827, according to Whitney, these mines yielded 42,000 tons of lead, thereby so reducing the price that the miners entered into a mutual agreement to work during one-half the year only. These deposits are not in veins, and are compared by Le Play to an immense amygdaloid, in which the paste is limestone and the amygdules galena. The limestone is of Silurian age.

From the nature of the deposits, it is evident that so large a production could not be continuously kept up; and from 1827, when it attained its highest point, the falling-off was rapid; at the present time these mines are considerably less productive than formerly.

At present the principal lead-producing districts in Spain are Linares, in the province of Jaen, Cartagena and Mazaron in the province of Murcia, and various mining fields in the province of Almeria.[1] Lead ore is also produced in the provinces of Granada, Estremadura, Badajoz, and to a small extent in some of the northern provinces. The Linares district may be said to embrace the neighbouring districts of Bailen, Baños, Vilches, La Carolina, and Santa Elena, an area of about 84 square miles; but two-thirds of the production comes from Linares proper, an area of not more than 20 square miles. The veins in the Linares district occur principally in granite, those in the neighbouring districts in clay-slate. The ore obtained is galena, which is dressed up to 75 or 78 per cent. of lead, and contains from 6 to 10 ounces of silver per ton of ore. So-called carbonates are also produced, which, although containing a certain amount of earthy carbonate of lead, depend principally for their value on partially decomposed galena; they are chiefly obtained from the smaller mines insufficiently supplied with washing apparatus, and contain from 45 to 60 per cent. of lead. A certain amount of slag is also obtained from washing the heaps left by the ancients; their tenure in lead, when sold, being from 30 to 50 per cent.

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[1] Mr. T. Sopwith has kindly furnished much information relative to the production of pig-lead in Spain.

The production of the entire Linares district is not less than 90,000 tons of galena per annum, to which may be added 10,000 tons of carbonates and 10,000 tons of slags. The carbonates and slags are nearly all smelted at Cartagena.

The Cartagena district may be said to include Herrerias, Porman, Mazaron, Aguilas, Cabo de Gato, Sierra Almagrera, and others. When Cartagena stood at the head of the list of Spanish lead-producing districts slags were the principal source of the metal. Including what is received from Linares, about 30,000 tons of pig-lead are now produced annually, principally from earthy carbonates found in layers in the limestone rock. Excluding imports from Linares, it is not probable that more than 8,000 to 10,000 tons of galena are smelted. The carbonates of, say 20 per cent. produce, are reduced, partly in numberless little smelt-mills scattered all over the district, and also at Escombrera, three miles from Cartagena, as well as at Cartagena itself, where there are large smelting establishments.

Almeria, including Motril, Guadix, Sierra de Baza, Solana, Berja, &c., has greatly diminished its production within the last ten years. It formerly produced nearly 80,000 tons of ore annually; its present production may be 10,000 to 15,000 tons of galena, yielding about 78 per cent. of lead.

The three districts named are the only ones in which mines are worked on a large scale; an additional 7,000 or 8,000 tons will cover all the remaining production of Spain. The total production of the country is believed to be nearly as follows:—

| | Tons of Pig-lead. |
|---|---|
| Linares, exclusive of slags and carbonates . . | 60,000 |
| Cartagena, including galena slags and carbonates from Linares . . . . . . . . | 30,000 |
| Almeria . . . . . . . . . . | 30,000 |
| Other districts, 8,000 tons at 76 per cent. (or 70 per cent. nett) . . . . . . . | 5,600 |
| | 125,600 |

Say, 120,000 tons of pig-lead annually.

It is not probable that the production will continue at this figure, as the low prices of lead which have ruled from 1880 to 1885 do not enable lead ores which do not contain a considerable amount of silver to be worked with profit. The production of late years has been kept up in a large measure by attacking the reserves of ore left in former times.

Of the 60,000 tons raised at Linares (nearly one half of the production of Spain) about one half is smelted in the district, and the remainder is carried to Cartagena and Almeria, and there smelted with the argentiferous ores and fluxes found on the south coast, or to Germany. The latter country takes from 10,000 to 12,000 tons of lead ore from Spain annually.

In the Piedmontese Alps the mines of Pesey and Macot have been worked during the last 150 years, and, together with that of Saint Jean de Maurienne, produced, according to Burat in 1846, about 250 tons of lead and 19,000 ounces of silver annually. In the island of Sardinia there are numerous mines, producing considerable quantities of lead and silver. Thirty-nine different mines were in operation in 1872, their annual production of lead being about 14,000 tons, containing 364,500 ounces of silver; in addition to this, Sardinia afforded about 900 tons of lead and 15,000 ounces of silver, obtained by smelting Roman and other slags.

The annual produce of the kingdom of Italy is estimated at 37,500 tons of argentiferous lead ore.

In 1881 Greece exported 11,700 tons of pig-lead, of which a large proportion was obtained from the ancient slags of Laurium.

The most important mines of argentiferous galena in France are those of Pontgibaud in the Puy-de-Dôme. The ore, which is much mixed with silica, contains a large amount of silver, and is smelted in blast-furnaces, after having been subjected to a preliminary roasting. The annual yield of the Pontgibaud mines is about 1,400 tons of lead and 145,000 ounces of silver; the total annual yield of lead ores in France is estimated at 14,000 tons, containing 185,000 ounces of silver.

The principal lead deposits of the United States of America were formerly situated in the Mississippi Valley, and were embraced in the States of Illinois, Wisconsin, Iowa, and Missouri.

Attention was first directed to the lead deposits of America by the famous expedition of Le Sueur, who, in his voyages up the Mississippi in 1700 and 1701, noticed many lead veins along its banks. The mines of Missouri had, however, been worked for some time before any further attention was given to the comparatively remote region of the Upper Mississippi; but in the year 1788 a Frenchman, Julien Dubuque, who had settled in the district, commenced mines on the western bank of the river on a tract of land which includes the now flourishing town of Dubuque. The principal mining centres are Galena in Illinois; Mineral Point, in Wisconsin; and Dubuque, in Iowa. The lead is almost exclusively found in a certain portion of the Lower Silurian formation, and there are no deposits in the Valley of the Mississippi which can be considered as coming under the head of true veins. The lead of this region is extremely poor in silver, seldom containing much above one ounce to the ton.

Numerous true veins containing galena occur in the Eastern States of North America; but although these afford ores which are richer in silver than those of the deposits of the Mississippi Valley, they are but little worked, and their yield of lead is comparatively small.

Of late years large deposits of argentiferous lead ores have been discovered in Utah, Nevada, Colorado, Montana, and Idaho; and a very large portion of the lead now annually produced in the United States is obtained from those regions.

The present annual production of lead in the United States is 133,000 tons, and the total production of the world is probably about 450,000 tons.

## ASSAY OF LEAD ORES.

The ores of lead may, for the purposes of assay, be divided into two classes.

The first class comprehends all ores of lead and other plumbiferous substances which contain neither sulphur nor arsenic, or in which these bodies are present in small proportion only.

The second class comprises sulphide of lead, or galena, together with all lead ores containing either arsenic, phosphorus, or sulphur.

From the facility with which lead is volatilized when strongly heated it is necessary to conduct the assay of its ores at a moderate temperature, as a notable quantity of the reduced metal would otherwise be driven off in the state of vapour.

The furnace best adapted for making lead assays is constructed similarly to that used for the assay of the ores of iron and copper, but may be of somewhat smaller dimensions. For this purpose, the internal cavity for the reception of fuel should be about 9 inches square, and the height of the throat from the fire-bars about 13 inches.

A furnace of this kind should be connected with a chimney of at least 20 feet in height. Moderately hard coke, broken into pieces of about the size of hens' eggs, is the most convenient fuel.

**ASSAY OF ORES OF THE FIRST CLASS.**—The assay of ores belonging to this class is a simple operation, care being only required that a sufficient amount of carbonaceous matter be added in order to effect the complete reduction of the metal, while such fluxes are supplied as will afford, by combining with the siliceous and earthy matters present, a liquid and readily fusible slag. The mineral to be assayed is first pounded in an iron mortar and passed through a sieve of wire-gauze. Those portions which remain on the meshes are again crushed until the whole has been passed through, since, if this were not attended to, a fair sample of the ore could not be obtained; as the more sterile portions, being usually the hardest, are the last to become sufficiently reduced in size.

When the ore has been properly ground, 400 grains may be weighed out and well mixed with 600 grains of dry sodium carbonate, and from

40 to 50 grains of powdered charcoal, according to the supposed richness of the mineral.

This is now introduced into an earthen crucible of such a size as to be not more than one-half filled by the mixture, and on the top of the whole is placed a thin layer, either of sodium carbonate or of common salt. The crucible with its contents is then placed in the furnace and gently heated, care being taken so to moderate the temperature that the mixture of ore and flux, which soon begins to soften and enter into ebullition, may not swell up and flow over the sides. If the effervescence becomes too strong, it must be checked by partially removing the crucible from the fire, and by a due regulation of the draught by the damper.

When the boiling has subsided and gas is no longer given off, the heat is again raised during a few minutes and the assay completed. During the process of reducing the metallic oxide or carbonate the heat should not exceed dull redness; but in order to complete the operation and render the slags sufficiently liquid to admit of the accumulation of the lead in one button at the bottom of the crucible the temperature must subsequently be increased to bright redness.

When the contents of the pot have been reduced to a state of tranquil fusion, it must, by the aid of proper tongs, be removed from the fire, and the assay poured into an iron mould; or, after having been tapped gently against some hard body, to collect the lead in a single globule, it may be set aside to cool. When the operation has been successfully performed, the cooled slag will present a smooth concave surface, with a distinct vitreous lustre. As soon as the crucible has become sufficiently cold it is broken, and the button of lead extracted. To remove from the metal, obtained either from the mould or by breaking the crucible, the particles of adhering slag, the button is hammered on an anvil, and afterwards washed and rubbed with a hard brush. If any portions of the slag adhere so firmly as not to admit of being easily removed by mechanical means, it may, in most instances, be separated by placing the button for a short time in a little dilute sulphuric acid, by which the slag is dissolved, whilst the metallic button remains unaffected. When the ore has been properly fluxed and a liquid slag obtained, the whole of the metallic lead will have collected in one mass at the bottom of the pot.

Instead of employing sodium carbonate and powdered charcoal, the substance to be assayed may be fused with twice its weight of black flux, and the mixture slightly covered by a thin layer of borax. Very good results are also obtained by mixing together—

400 grains of plumbiferous matter,
500 ,, carbonate of sodium, and
200 ,, crude tartar.

These ingredients, after being perfectly incorporated, are placed in an earthen crucible and covered with a thin layer of borax.

The three foregoing methods yield equally good results, and afford slags containing but a very small portion of lead.

The assay of very rich lead products belonging to this class may be conducted without the use of any kind of flux, as, when heated to redness in a lined crucible, they are readily and completely reduced. It is, however, desirable in all cases to add about 10 per cent. of sodium carbonate, by which the adherence of any metallic globules to the charcoal lining is prevented. This method, although requiring a longer time than those already described, does not, even when rich ores are operated on, afford more satisfactory results.

When the substance operated on contains, in addition to lead, other metals—such as copper, silver, tin, or antimony—the button obtained will retain a greater or less proportion of these metals. If zinc be present in the ore, traces only of that metal will be discovered in the resulting button of lead. For commercial purposes the button of lead is seldom subjected to chemical examination, as the purity of the metal is usually judged of in accordance with its colour and softness; but when a more accurate knowledge of its constituents is required, it must be made to undergo the usual routine of analysis. The carbonates of lead are the only important minerals belonging to this class.

ASSAY OF ORES OF THE SECOND CLASS.—This class not only comprehends galena, which is the most common and abundant ore of lead, but also comprises the sulphides resulting from various metallurgical processes, as well as the sulphates, phosphates, and arsenates of that metal.

The assay of galena is variously conducted, but one of the following methods is most commonly employed for commercial purposes.

The ore to be examined, after having been properly crushed and sifted, is fused, either—

1. With sodium carbonate or black flux;
2. With metallic iron;
3. With sodium carbonate or black flux and iron; or,
4. With a mixture of nitre and sodium carbonate.

*First Method: Fusion with an Alkaline Flux.*—This operation is conducted in an earthen crucible, which is to be left uncovered until its contents have been reduced to a state of tranquil fusion.

The powdered ore, after being mixed with three times its weight of dry sodium carbonate, is slowly and gradually heated in an ordinary assay furnace until the mixture has become perfectly liquid, when the crucible is removed from the fire, and, after having been gently tapped to collect any globules of metal which may be in suspension in the slag, is set aside to cool. When cold, the crucible may be broken, and a button of metallic lead, which must be cleaned and weighed, will be found at the bottom.

Instead of sodium carbonate, either carbonate of potassium or black flux may be used; but when the last-named substance is employed a little longer time is necessary for the complete fusion of the mixture.

Every 100 parts of pure galena will, by this method, afford from 75 to 77 parts of metallic lead, indicating a loss of from 7 to 9 per cent. on the contents of the ore.

Some of the older assayers were in the habit of partially expelling the sulphur by roasting, and afterwards reducing the resulting oxide with about its own weight of black flux.

This process, from the easy fusibility of the sulphides and oxides of lead, requires very careful manipulation, and, at best, the results obtained are far from satisfactory. Pure galena, by this method, can rarely be made to afford above 70 per cent. of metallic lead.

*Second Method: Fusion with Metallic Iron.*—This process depends on the circumstance, that when galena is fused in contact with metallic iron, that metal becomes converted into ferrous sulphide, whilst the lead originally combined with the sulphur is liberated. The amount of iron actually required for the decomposition of pure sulphide is about 22·5 per cent.; but it is advantageous, in practice, to add a small excess of this metal, and 30 parts of iron to every 100 parts of galena may therefore be employed.

The iron used should be either in the form of small nails or in that of wire cut into short pieces. This mixture of ore and metallic iron is placed in a clay crucible, of which it should not fill above two-thirds the capacity, and is covered with a thin layer of either sodium carbonate or borax. The crucible with its contents is afterwards heated to full redness, by which a well-fused and perfectly liquid slag is produced. When the contents of the pot are in a state of tranquil fusion, it is removed from the furnace and allowed to cool. It is then broken, and at the bottom will be found a button, which at first sight appears to have throughout a uniform composition, but on being struck with a hammer separates into two distinct parts. The upper portion consists of sulphide of iron, which crumbles under the hammer and is readily removed; while the lower part is a button of malleable lead, which must be cleaned and weighed. This process affords, from pure galena, about 78 per cent. of metallic lead. The loss appears to arise principally from the volatility of galena, which begins to be driven off at a lower temperature than is required for its decomposition by iron.

In North Wales this method of assay is sometimes conducted in a manner somewhat different from that described. Instead of adding finely divided iron to the ore, the pounded mineral is heated, without the addition of any kind of flux, in a ladle made of that metal. This ladle or *dish* is formed out of a thick piece of plate-iron, and is provided with a lip, by which the reduced metal is poured off, and with four projecting corner-pieces, which afford a holdfast for the tongs by which it is removed from the fire. The ore to be operated on is first coarsely powdered and well mixed, so as to ensure a fair sample. Eight ounces are now weighed out and placed in the dish, which is covered with a lid of sheet-iron, and gently heated in the fire of a smith's forge until the ore ceases to decrepitate. The temperature is then raised to full redness, and at the expiration of about five minutes the decomposition of the sulphide will be completed. At this point the dish is removed from the fire and the reduced lead poured into a cast-iron mould, while the slags and sulphide of iron formed are kept back in the dish by a piece of wood held behind the spout for that purpose. The dish, together with the slags and iron sulphide, is afterwards again placed on the fire and heated to bright redness during another five minutes, by which the last portions of metallic lead adhering to the slags are obtained. The contents of the dish are now thrown away, as not containing any lead, while the metal which has been run off is carefully weighed. This apparently rude method affords, in experienced hands, remarkably good results, which are likewise considered to approach very nearly to the practical returns obtained during metallurgical treatment on the large scale. By this process, pure galena yields from 79 to 83 per cent. of lead; but with the poorer varieties of ore, such as those obtained from some of the Cornish mines, satisfactory results could not be obtained, since, from the infusibility of the associated gangue, &c., a considerable amount of lead would be retained in the slag. The ladles used for this purpose are rudely made of plate-iron, which, if about one-fourth of an inch in thickness, will last during three or four separate trials.

*Third Method: Fusion with Sodium Carbonate or Black Flux and Metallic Iron.*—When galena and sodium carbonate are fused together out of contact with the air, a large proportion of the lead is liberated in the metallic form, but the slag retains a certain amount of that metal in the state of a double sulphide of lead and the alkali-metal.

If iron be now introduced, the sulphide of lead contained in the slag will be decomposed; metallic lead is liberated, and the slag contains a double sulphide, in which iron will have replaced the lead formerly present. The earthy and siliceous matters constituting the gangue are also dissolved in the slag without, to any great extent, impairing its fluidity.

The quantity of black flux or sodium carbonate employed varies with the richness of the ore operated on; but even for the poorest varieties two parts of the alkaline reagent will be found sufficient. The iron, which is merely used to separate that portion of the lead which has been dissolved by the alkali in the state of sulphide, need not be present in sufficient amount to effect the reduction of the whole of the lead contained in the ore treated. Two parts of black flux or sodium carbonate, and from 10 to 15 per cent. of metallic iron, either in the state of filings or in the form of nails, will be found a convenient quantity for this purpose.

When the fusion is made with black flux, and the iron is in the state of filings, it will be proper not to add too large an excess, especially if the assay be conducted at a high temperature, as in that case the resulting button of lead will contain iron. If, however, carbonate of sodium be employed, the addition of a small excess of iron is attended with advantage, as it ensures the complete desulphurization of the galena without affecting the purity of the resulting lead.

Iron filings, when employed for this purpose, are liable to become mechanically intermixed with the lead, and thereby, to a certain extent, falsify the results. This inconvenience is obviated by the use of small iron nails, which are corroded only on the outside, and, at the termination of the assay, are found fixed in the upper surface of the button, from which they can without difficulty be separated. Pure galena, when thus treated, yields from 75 to 78 per cent. of metallic lead.

The following process is a slight modification of that long employed at the École des Mines, Paris. Two earthen crucibles are prepared by smearing their insides with black-lead, such as that used for stove polishing, and in each of these are placed, with their heads downwards, three or four large flooring-nails. Mix the ore to be assayed with its own weight of sodium carbonate, and, after having placed it in the pots, press it tightly down about the nails. On the top of this, place about half an ounce of common salt, and above it an amount of dried borax equivalent to the weight of the ore operated on. The whole is now introduced into the furnace and gradually heated to redness; at the expiration of ten minutes the temperature is increased to bright redness, at which it is kept for another ten minutes, when the flux will be fused and will present a perfectly smooth surface. When this has taken place, the pot is removed from the fire, and the nails are separately withdrawn by the use of a pair of tongs, care being taken to well wash each in the fluid slag until perfectly freed from adhering lead. When the nails have all been withdrawn, the pot is gently tapped, to collect the metal into one button, and then laid aside to cool; after which it is broken, and the button of lead removed and cleaned in the usual way. The result is then verified by a second assay, made in the other pot.

When carefully conducted, this process is said to afford, from pure galena, from 84 to 84¾ per cent. of metallic lead. It is, however, liable to the objection, that the lead produced sometimes contains fragments of iron, arising from the nails being most energetically acted on at the point of contact between the flux and the galena, which, when the slag becomes fused, occupies the lower portion of the crucible; by this means portions of iron may become detached and adhere firmly to the button, from which there is sometimes difficulty in removing them. The writer has never obtained by this process above 83 per cent. of metal from the purest specimens of galena, but, notwithstanding, it affords results sufficiently accurate for many commercial purposes. In all cases where earthen crucibles are employed the assay may be poured into an iron mould, instead of breaking the pot; the crucible can be thus preserved and used for a second assay.

In place of adding metallic iron to the mixture of ore and flux introduced into the crucible, it is better that the pot itself should be made of that metal.

For this purpose, a piece of half-inch plate-iron, of good quality, is turned up in the form of a crucible and carefully welded at the edges; the bottom is closed by a thick iron rivet, which is securely welded to the sides, and the whole is then finished up with a light hammer on a properly formed mandrel. The crucible, when finished, should have the form represented in fig. 169. To make an assay in a crucible of this description it is first heated to dull redness, and, when sufficiently hot, the powdered ore, intimately mixed with about its own weight of a mixture of two parts of sodium carbonate and one part of dried borax, is introduced by means of a long copper scoop of the form represented in fig. 170, and the crucible, which, for the introduction of the mixture, has been removed from the fire, is immediately replaced. The heat is now gradually raised to redness, during which time the contents become liquid and give off large quantities of gas. At the expiration of from ten to fifteen minutes the mixture will be observed to be in a state of tranquil fusion, when the pot is partially removed from the fire and its contents are briskly stirred with an iron rod. Any matters adhering to its sides are also scraped down to the bottom of the pot, which, after being again placed in a hot part of the furnace, is closed with an earthen cover and heated during three or four minutes to somewhat bright redness. The crucible is then seized by strong bent tongs on that part of the edge which is opposite the projecting lip, *a* (fig. 169), and, after being removed from the fire, its contents are rapidly poured into a cast-iron mould, fig. 171. Another form of mould is represented in fig. 172. The sides of the pot are now carefully scraped down with a chisel-edged iron bar, and the adhering particles of slag and of metallic lead are added to the portion first obtained, by sharply striking the edge of the pot, firmly held in the jaws of the tongs, against the top of the cast-iron mould. When sufficiently cold the contents of the mould are removed, and the button of lead, after having been separated from the adhering slag, is carefully cleaned

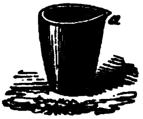

Fig. 169.

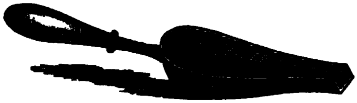

Fig. 170.

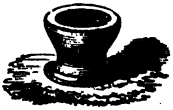

Fig. 171.

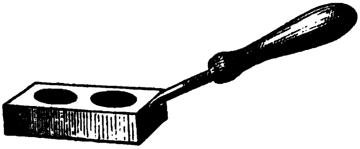

Fig. 172.

and weighed. By this process pure galena yields, on an average, 84 per cent. of metallic lead, free from iron, and perfectly malleable. This method of assaying is that which is in almost universal use in lead-smelting establishments, and has the advantage of yielding good results with all the ores belonging to the second class. A larger amount of lead is, however, obtained by assay than can be produced from the same ores in the large way; and on this account the smelter makes an allowance in accordance with the nature of the ore treated.

Instead of using iron pots, or adding metallic iron to the ores, they may be fused with a mixture of black flux and oxide of iron; this method, however, does not afford satisfactory results, and is not so convenient as the processes in which an iron pot is employed.

*Fourth Method : Fusion with Sodium Carbonate and Nitre.*—When galena is treated with nitrate of potassium, the whole of its sulphur is converted into $SO_3$, before any portion of the lead is oxidized; and it consequently follows, that if a suitable amount of nitre were employed the desulphurization of the metal would be completely effected, and the whole of the lead obtained in the metallic state. To prevent any loss which might arise from the deflagration which takes place, the ore is mixed with twice its weight of sodium carbonate, and to this, in accordance with the richness of the ore, is added from 30 to 35 per cent. of nitrate of potassium. When too large a quantity of nitre is employed, a portion of the metal will be oxidized and remain in the slag, causing a corresponding deficiency in the weight of the button obtained; therefore, as the proper amount can only be ascertained by repeated experiments, this process is but ill adapted for the assay of ores for lead.

When, on the contrary, an ore contains silver, and that metal only is to be estimated, regardless of the amount of lead present, this process may sometimes be employed with considerable advantage, although it is generally less to be recommended than fusion with sodium carbonate and borax in a wrought-iron pot, as before described. The assay with nitre is very easily conducted. The fusion takes place readily, and without bubbling, and the slag, which is very liquid, contains no metallic shot. In conducting this operation, the amount of nitre should, if possible, be so arranged as to afford the largest quantity of metallic lead; but it is of importance that enough to decompose the whole of the sulphide should be employed, since if the slag retain any unoxidized sulphur-compound a loss of silver may be experienced. When, on the other hand, too large a proportion of nitre has been used, the accuracy of the silver estimations will not be impaired, as lead alone is oxidized, and the whole of the silver will be contained in the metallic button.

ASSAY OF GALENA CONTAINING ANTIMONY.—Many of the ores of lead contain a certain proportion of antimony, and from such minerals, according to Berthier, the assayer can obtain, at will, either pure lead, or a mixture of the two metals in the form of an alloy.

To extract pure lead the ore may be fused in an open crucible, with three times its weight of sodium carbonate, when the lead will be liberated in the metallic form, while the antimony, becoming oxidized, unites with the alkali and remains wholly in the slag. The presence of antimony in the slag also prevents its retaining any portion of the lead, and from this cause a tolerably exact separation of the two metals is obtained.

When, in addition to lead and antimony, an ore contains silver, it should be assayed by being heated with a mixture of sodium carbonate and nitre, by which the antimony will be oxidized and retained in the slag, while the lead and silver are obtained in the state of an alloy. In this case it is only necessary to add a sufficient amount of nitre to effect the total decomposition of the sulphides, as even when the whole of the antimony and a portion of the lead have been oxidized no loss of silver is experienced; but when, on the contrary, the slags contain undecomposed sulphide of antimony, a small portion of the silver is lost.

When it is required to reduce at the same time both the antimony and lead, the ore may be fused with a mixture of borax and sodium carbonate in an iron pot, in accordance with the process before described for assaying lead.

Sulphate of lead is readily reduced by simple fusion with black flux and sodium carbonate in an earthen crucible; but when phosphates and arsenates are present in ores of this metal their assay should be conducted in an iron pot, with a proper admixture of sodium carbonate and fused borax.

Various methods have been proposed for the estimation of lead by standardized solutions, but none of them appears to have been successfully applied to the assaying of lead ores.

## ESTIMATION OF SILVER IN LEAD ORES.

From the amount of silver contained in many varieties of galena and other ores of lead it becomes necessary, in order to determine their commercial values, to ascertain with great exactitude their yield of this metal.

For this purpose, the button of lead, from any of the assay processes already described, is subjected to the operation known as cupellation. This process is founded on the circumstance that when lead containing silver is melted and exposed to a current of air at a red heat, oxide of lead is formed, which melts and covers the metal; but if the operation is so conducted that the melted oxide or *litharge* is removed as it forms, the action continues until the whole of the lead is oxidized, and the silver is left in the metallic state. This is done in the assay furnace by the use of shallow thick-bottomed dishes made of bone-ash, a substance chemically indifferent to oxide of lead and impermeable to the metal, but susceptible of absorbing melted litharge in quantity.

These dishes, called *cupels*, figs. 173 and 174, the first of which is a section, are made of bone-ash, slightly moistened with water, and tightly consolidated by pressure in an iron mould.

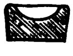

Fig. 173.

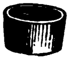

Fig. 174.

A convenient furnace for cupellation is represented in figs. 175 and 176, of which the first is an elevation and the second a vertical section. The material of which this furnace is made is wrought-iron, lined with fire-tiles, as shown in the drawing. The *muffle*, *m*, is a small D-shaped retort of fire-clay, closed at one of its extremities only, and sometimes furnished with holes or perpendicular openings in the sides and end, in order to allow of a free circulation of air. Fig. 177 represents such a muffle before its introduction into the furnace. When fixed, it is so arranged that while one of its extremities is supported by a proper shelf the other corresponds with the opening *d′*, to the sides of which it is luted by a little moistened fire-clay. This position of the muffle allows of its being heated on every side by a supply of ignited fuel, while air entering by the opening, *d′*,

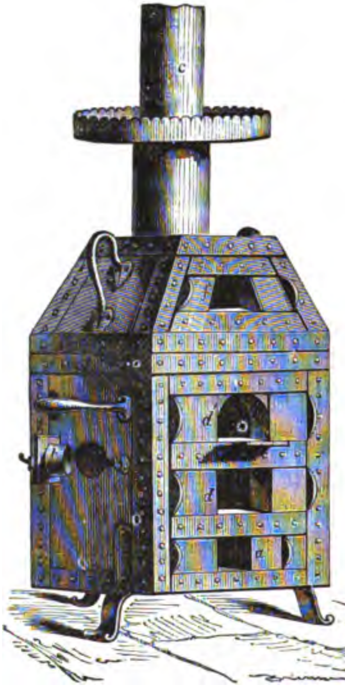

Fig. 175.

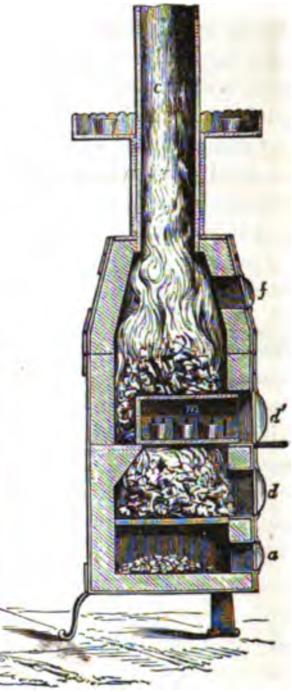

Fig. 176.

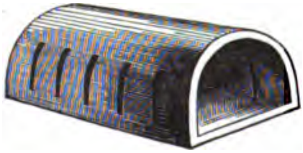

Fig. 177.

circulates through the interior. The cavity of the muffle is in this way traversed by a highly oxidizing current of air, and the draught of the furnace is increased by the addition of a sheet-iron chimney, *c.* To light this apparatus a little ignited coke or charcoal is introduced through the opening, *d*; the furnace is afterwards filled with the same fuel, and all the openings, excepting the ash-pit, *a*, are closed by their proper slides. When the charcoal or coke is properly ignited, and the muffle has become red hot, six or eight cupels, which have been warming on the ledge around the chimney, are taken by the tongs, fig. 178, and placed on the floor of the muffle, which, to prevent its becoming corroded, if any lead should be spilt upon it, is previously covered with a thin layer of ground bone-ash.

Fig. 178.

The opening, *d'*, is again closed, so as to prevent the introduction of cold air, and the cupels are raised to the temperature of the muffle itself. When this is the case, the door is again removed, and into each of the cupels is introduced, by a pair of slender steel tongs, a button of the lead to be assayed. The door is now, a second time, closed during a few minutes, to facilitate the fusion of the metal, and on its removal each of the cupels is found to contain a bright convex metallic disc, in which state the assays are said to be *uncovered.* The air thus admitted rapidly converts the lead into litharge, which, as fast as it is produced, is absorbed by the bone-ash of the cupel, and at the same time there arises a white vapour, which partially fills the muffle, and either escapes at the mouth, or is carried off through the openings in its sides. An annular stain, which gradually extends, is at the same time formed around the metal, and penetrates into the substance of the cupel, in proportion as the metallic globule itself diminishes in size. When nearly the whole of the lead has been thus converted into litharge, and has been absorbed, the remaining bead of rich alloy appears to become agitated by a circular movement, by which it seems to be made to revolve. At this stage of the operation the agitation will be observed suddenly to cease, and the button, after having for a moment emitted a bright flash of light, becomes white and immovable.

This phenomenon is called the *brightening* of the assay, and a button of silver remains on the cupel.

If the cupel were at this period abruptly removed from the muffle the metallic globule would be liable to *sprout* or *vegetate*, by which a portion of it is not unfrequently thrown off and lost, while its surface is covered by numerous arborescent asperities. To prevent this from taking place, and to guard against the loss of metal which might ensue, the cupel on which the button of silver has brightened may be covered by another, kept red hot for that purpose. The two are subsequently withdrawn together, and allowed to remain on the ledge before the muffle until the metal has become solidified, when the upper cupel may be removed, and the globule of silver detached and weighed.

From the circumstance that silver is sensibly volatile at very elevated temperatures, it becomes necessary to make cupellations for this metal at the lowest heat by which the absorption of litharge can be determined. If, however, the cupel be not sufficiently hot, an annular incrustation of crystalline litharge will accumulate around its edges, and if, at this point, the fire be not immediately attended to, the deposit of oxide spreads over the whole surface of the metal, and its further oxidation is arrested.

The temperature best suited for this operation is obtained when the muffle and the enclosed cupels are at a full blood-red heat, and the vapours which arise from the alloy curl gradually away and are promptly removed by the draught. When the muffle is heated almost to whiteness, and the vapours rise to the crown of the arch, the temperature is too high, and when, on the contrary, the fumes lie over the bottom, and the sides of the openings in the muffle begin to blacken, more fuel must be added through the door, *f*, and the heat gradually raised. When the operation is conducted at a proper temperature, the cupel should be of a full-red colour, and the fused alloy bright and convex. At the commencement of the operation the heat must be a little raised, for the purpose of fusing and uncovering the button, and just before the globule is about to brighten a slight elevation of temperature is again advantageous, but if a good red-heat has been kept up during the working of the assay, this is not necessary.

The success of the operation is likewise influenced by the force of the draught passing through the muffle. When the current is too rapid, the cupel becomes cooled, and the lead is oxidized with greater rapidity than it should be. In this case the litharge produced is sometimes not absorbed by the test as fast as it is generated, and consequently the surface of the alloy is covered by a layer of lead oxide, by which it may ultimately become protected from further oxidation. When, on the contrary, the current is too feeble, the assay remains too long in the muffle.

If an assay has been properly conducted, the button of silver is round, bright, and smooth on its upper surface, and beneath should be crystalline, and of a dead-white colour; it is easily detached from the cupel, and readily freed from adhering litharge. This globule is now removed and squeezed between the jaws of a pair of pliers, by which the oxide of lead, which attaches itself to it, becomes pulverized, and is easily removed by a small brush made of stiff hogs' bristles. The cleaned metallic bead is afterwards weighed in a balance capable of turning with $\frac{1}{1.000}$ grain.

The fuel employed in the furnace above described, after it has once got into steady action, consists of small pieces of hard coke. When, in addition to silver, ores of lead likewise contain gold, the button remaining on the cupel will consist of an alloy of these metals, the separation and estimation of which will be treated of under the head of *parting* when describing the assay of gold ores.

In laboratories in which assays of gold and silver are constantly made, the furnace above described, which has holes in two opposite sides closed by stoppers, *t* (and can consequently be used as a tube-furnace), is inconvenient, on account of its small size, &c. For this reason, either fire-clay furnaces bound with iron, or stationary furnaces, forming part of the building of the laboratory, are frequently employed.

Silver contained in lead or ores is reported in troy ounces, dwts., and grains, per ton of 2,240 lbs.; and in metrical countries in *grammes* per *tonne* (1,000 kilogrammes).

Table showing the Amount of Silver to the Ton of Ore, corresponding to the Weight obtained from 400 Grains.

| If 400 Grains of Ore give Fine Metal | One Ton of Ore will yield | | | If 400 Grains of Ore give Fine Metal | One Ton of Ore will yield | | |
|---|---|---|---|---|---|---|---|
| gr. | oz. | dwt. | gr. | gr. | oz. | dwt. | gr. |
| ·001 | 0 | 1 | 15 | ·200 | 16 | 6 | 16 |
| ·002 | 0 | 3 | 6 | ·300 | 24 | 10 | 0 |
| ·003 | 0 | 4 | 21 | ·400 | 32 | 13 | 8 |
| ·004 | 0 | 6 | 12 | ·500 | 40 | 16 | 16 |
| ·005 | 0 | 8 | 4 | ·600 | 49 | 0 | 0 |
| ·006 | 0 | 9 | 19 | ·700 | 57 | 3 | 8 |
| ·007 | 0 | 11 | 10 | ·800 | 65 | 6 | 16 |
| ·008 | 0 | 13 | 1 | ·900 | 73 | 10 | 0 |
| ·009 | 0 | 14 | 16 | 1·000 | 81 | 13 | 8 |
| ·010 | 0 | 16 | 8 | 2·000 | 163 | 6 | 16 |
| ·020 | 1 | 12 | 16 | 3·000 | 245 | 0 | 0 |
| ·030 | 2 | 9 | 0 | 4·000 | 326 | 13 | 8 |
| ·040 | 3 | 5 | 8 | 5·000 | 408 | 6 | 16 |
| ·050 | 4 | 1 | 16 | 6·000 | 490 | 0 | 0 |
| ·060 | 4 | 18 | 0 | 7·000 | 571 | 13 | 8 |
| ·070 | 5 | 14 | 8 | 8·000 | 653 | 6 | 16 |
| ·080 | 6 | 10 | 16 | 9·000 | 735 | 0 | 0 |
| ·090 | 7 | 7 | 0 | 10·000 | 816 | 13 | 8 |
| ·100 | 8 | 3 | 8 | | | | |

The bone-ash employed for making cupels is first passed through a sieve of wire gauze, and afterwards mixed with water until sufficiently moistened to retain the mark of the fingers, when a handful is taken up and tightly squeezed. To cause the cupels when made to be sufficiently firm and resistant, a little potassium carbonate is often added to the water employed for moistening the bone-ash. The amount of alkaline carbonate required for this purpose is exceedingly small, as a fragment of the size of a nut will be sufficient to add to a pint of water. Instead of water, some persons use sour beer, and in this case dispense with the use of alkali. The mould in which the cupels are formed, fig. 179, usually consists of a bevelled steel ring, *b*, and a die, *a*, made of the same metal, and often fitted with a wooden handle. To make the cupel, this cavity is nearly filled with the moistened bone-ash, which is first compressed slightly by the hand and afterwards with the die, which is tightly driven into the ring by the use of a wooden mallet, fig. 180. When sufficiently consolidated the die is withdrawn, and, by introducing a wooden cylinder exactly filling the aperture, the cupel is without difficulty removed. The use of the wooden cylinder is somewhat liable to crumble the edges of the cupel; and for this reason a loose iron plate, *c*, exactly fitting the bottom of the mould, is often introduced before the bone-ash is placed in its When this precaution is taken, the iron protects the bottom of the cupel, and enables the operator to use considerable force without injury to the edges of the newly made test. The iron plate has obviously to be replaced at each operation, and, with the cupel before it, is again forced out of the mould. In establishments in which a large number of cupels are used they are frequently made by the aid of a screw-press. When made, they are set aside to dry, and are then ready for use; but it is better, if possible, to keep cupels at least three or four weeks in a dry place before using them.

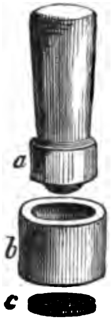

Fig. 179.

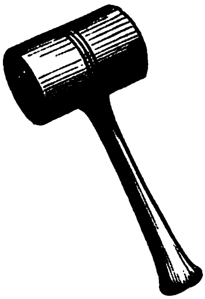

Fig. 180.

## METALLURGY OF LEAD.

By far the larger proportion, probably above nine-tenths of the lead annually produced, is obtained from galena, which before coming into the hands of the smelter is freed, by careful mechanical preparation, from the principal part of the earthy and siliceous veinstone with which it was originally associated. The metallurgical treatment of the ores of lead may be effected by either of the following general methods:—

Firstly.—By the method of *double decompositions*, or, as it is sometimes called, *method by reactions* (*Röstreactionsarbeit*). This is founded on the fact that when PbS is roasted at a gentle heat, it is partially converted into PbO and partly into $PbSO_4$, and that on raising the temperature the following reactions are set up:—

$$2PbO + PbS = 3Pb + SO_2$$
$$PbSO_4 + PbS = 2Pb + 2SO_2.$$

Secondly.—By roasting and subsequent reduction of the oxidized products, chiefly by carbonaceous matter (*Röstreductionsarbeit*).

Thirdly.—By the *precipitation process* (*Neiderschlagsarbeit*), in which desulphurization of the lead is effected by metallic iron.

Two of these processes, and sometimes even all of them, may be combined in the treatment of a lead ore.

Lead-smelting is conducted in three distinct varieties of apparatus, which may be thus classified :—

1. *Reverberatory furnaces.*
2. *Blast-furnaces.*
3. *Shallow hearths.*

## SMELTING IN REVERBERATORY FURNACES.

FLINTSHIRE PROCESS.—This, which is essentially a method by double decompositions, comprehends the following succession of operations :—

Calcination at a low heat in order to produce a certain amount of oxide and sulphate of lead.

So raising the temperature as to determine the fusion of the calcined products, thereby causing the liberation of a large amount of lead by the reaction of its oxide and sulphate upon unchanged sulphide.

The incorporation of the residue with lime, and the reduction of the unchanged sulphide, with formation of a larger amount of oxidized lead-compounds than is required to effect the decomposition of the sulphide of lead.

Tapping the reduced lead, and the subsequent removal of the pasty grey slag by drawing.

The furnace employed for the reduction of lead ores by the Flintshire process varies somewhat in its construction and dimensions; the length of the hearth is usually about 11 feet, and its width 9 feet, and under this is an arched vault, extending the whole length of the bed to the fire-bridge. The hearth is made of slag, moulded into the proper form when in a plastic and semi-fused state, and has towards its centre a depression in which the fused metal accumulates, and at the bottom of which is situated the tap-hole. The fire-place is at one end, and before reaching the cavity of the furnace the flame has to pass over a fire-bridge, about 2 feet in width and from 12 to 14 inches below the arch; at the opposite extremity of the hearth are openings communicating with a flue in connection with a lofty chimney. The fuel is supplied through a door at one end of the fire-place, in addition to which the furnace is furnished with six working doors, about 9 inches by 12, protected by heavy cast-iron frames built into the brickwork, and closed by iron plates, which can be easily removed when required. The bottom of this furnace is made nearly level with the doors on one of the sides, but is inclined towards the other in such a way as to be from 18 to 20 inches below the middle door, where it communicates with the tap-hole A cast-iron tapping-pot is set in the ground beneath the tap-hole, and on the top of the furnace is a hopper, from which a fresh charge of ore is let down as soon as that which is being worked is withdrawn.

The charge of such a furnace is usually, in North Wales, 21 cwts., but as much as 24 cwts. is sometimes operated on at one time. In the neighbourhood of Newcastle the charges are usually smaller. As soon as the lead from the preceding operation has been tapped off, and the slags cleared out and removed, another charge of ore is dropped from the hopper upon the hearth, which is at this time barely red hot. This is spread in a tolerably even layer over the surface of the bed, care being taken to prevent any portion of it from falling into the deep part or *well* of the furnace. The charge is now frequently stirred during two hours, the supply of air being regulated by the partial opening or closing of the various doors, and the highest temperature, compatible with keeping the charge free from clotting, is maintained. At the expiration of this period the fire-bars are clinkered, and fresh fuel is thrown upon the grate. The charge now begins to assume a pasty condition, and any portions which may have run down towards the tap-hole are brought back to the higher parts of the hearth. The doors are then opened for the purpose of lowering the temperature, and as soon as the charge has acquired the consistency of stiff mortar, it is spread before the fire-bridge and upon the portions of the hearth furthest removed from the tap-hole.

This accomplished, the fire- and furnace-doors are again closed, the charge is run down as quickly as possible into the well, and a couple of shovelsful of slaked lime are thrown upon its surface and incorporated with the mass by stirring with a rake. By this means, the slags and unreduced ores are rendered sufficiently pasty to allow of their being again arranged or *set-up* on the sloping sides of the hearth, where they are calcined for about half an hour, and again run down; during this latter portion of the operation, a little coal-slack is sometimes thrown upon the charge. When the whole of the charge has again collected in the bottom, a further addition of slaked lime is made, and the slag is pushed off from the surface of the metal upon the inclined sides of the well, where it is allowed to drain. The lead is now tapped, and the *grey slag* is withdrawn in pasty lumps through the doors at the back of the furnace.[1]

The slags and mattes which accumulate on the surface of the tapping-

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[1] The following analysis of grey slag from a Flintshire furnace was made in Dr. Percy's laboratory by Mr. C. Tookey:—

| | |
|---|---|
| $PbS$ | 0·90 |
| $PbSO_4$ | 9·85 |
| $PbO$ | 48·87 |
| $ZnO$ | 7·52 |
| $CaO$ | 12·68 |
| $Al_2O_3$ | 3·01 |
| $Fe_2O_3$ | 2·86 |
| $SiO_2$ (combined) | 12·52 |
| Insoluble residue | 1·45 |
| | 99·66 |

pot contain a considerable amount of mechanically intermixed lead, to separate which a paddle is inserted, and the whole is well stirred. Some coal-slack is now thrown in and well worked up with the contents of the pot; this gives off a considerable amount of gas, which being ignited affords sufficient heat to liberate the metallic lead. The surface is then skimmed, the skimmings being at once thrown back into the furnace, and the lead is laded out into cast-iron moulds. After this the tap-hole is again opened, and the lead from the skimmings run out into the pot, where it is allowed to remain until that resulting from the next charge is withdrawn. The tap-hole is now closed, and another charge let down into the furnace from the hopper placed above it.

In Flintshire the lead ores treated by this process generally contain from 75 to 80 per cent. of lead, and a charge of such ore may be worked in from five and a half to six hours, yielding about 90 per cent. of the total quantity of the lead obtained; the remaining 10 per cent. being subsequently extracted from the slags and fume in a blast-furnace.

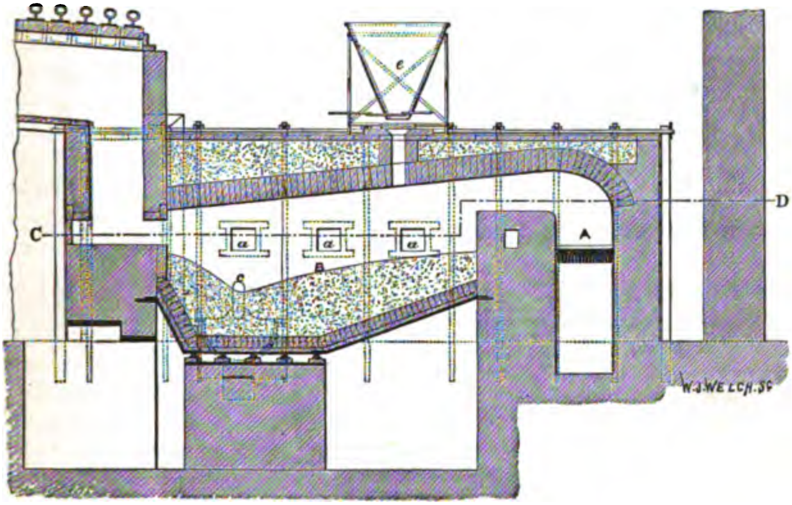

Fig. 181.—Smelting-Furnace, Couëron; longitudinal section.

**Modification of the Flintshire Processes at Couëron.**—The construction of the furnaces at Couëron, Loire Inférieure, France, will be understood by reference to the accompanying woodcuts. Fig. 181 is a longitudinal section; fig. 182, a horizontal section on the line C D; and fig. 183, a transverse section through the tapping-hole. The fire-place, A, is of the usual dimensions, and the hearth, B, instead of being on an arch, as is commonly the case in North Wales, is supported on iron bars, on which is laid a course of flat tiles. On these is placed a course of fire-bricks on edge, on which is arranged the usual slag bottom. The furnace has the ordinary number of working doors, *a*, with a fire-door, *b*, and tapping-hole, *c*; there is a small fire-place below the pot, *d*, to prevent the too rapid chilling of the lead during the process of lading into moulds. Although similar in form and dimensions to the ordinary Welsh furnace, it differs from it in one important particular, namely, in having the tapping-pot placed near the flue end instead of under the

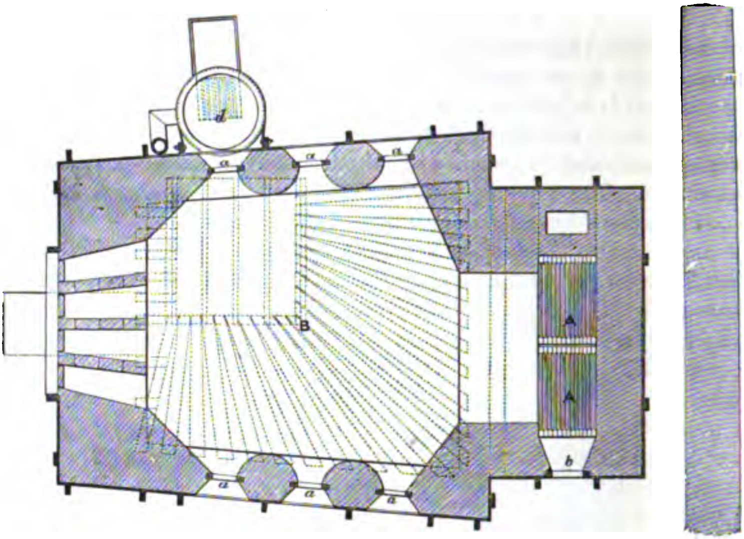

Fig. 182.—Smelting-Furnace, Couëron ; horizontal section on C D.

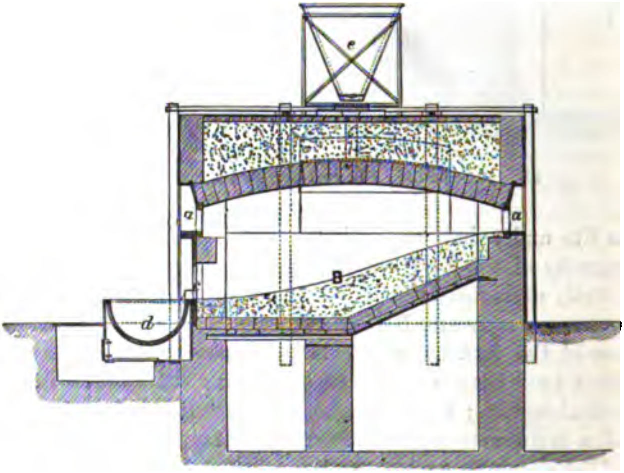

Fig. 183.—Smelting-Furnace, Couëron ; transverse section through tapping-hole.

middle door. This arrangement gives a larger surface to the hearth for roasting the charge, and permits of the lead being collected in the coolest part of the furnace, where it is least exposed to loss from volatilization.

The mode of working employed varies with the nature of the ores treated. This variation depends principally on the length of time required for roasting the ores before subjecting them to the higher temperature of reactions. Pure ores, especially those containing a notable proportion of carbonate or sulphate of lead, require very little previous roasting, whilst those containing blende, pyrites, &c., must be calcined for a considerable time before smelting.

In general, the ores treated are derived from Sardinia, and contain, on an average, 81 per cent. of lead and but little silver. They are tolerably free from impurities, being composed of galena associated with carbonate and sulphate of lead, and a calcareous gangue, in which are found small quantities of blende, calamine, ferric oxide, and silica. The cobbed ores, before delivery to the furnace, are ground between rolls, and passed through sieves of eight holes to the linear inch; a charge of ore weighs 27 cwts. Two men are employed at each furnace, one of whom, the chief, works on the fore side, and takes a leading part in the work to be done. In the treatment of rich ores, the ordinary method of working consists in roasting them in such a manner that upwards of one-half of the sulphide of lead present in the ore is converted into a mixture of oxide and sulphate.

On raising the heat in the furnace to bright redness, the oxide and sulphate, formed in roasting, react on the undecomposed sulphide in the charge, and produce metallic lead, sulphurous anhydride, and a residue of slag.

The process is thus divisible into two well-marked and distinct operations:—

*a.* Calcination or oxidation.
*b.* Smelting or reactions.

*a. Calcination.*—Supposing a charge to have been just worked off, and the residual slags withdrawn, the furnace will be empty and at a red heat. The damper having been lowered, the charge of ore in the hopper, *e*, is let down through the opening in the arch, and spread evenly over the furnace bottom by means of rabbles.

This done, the working-doors are closed, but the fire-door is left open and the fire damped with cinders, in order so to moderate the heat that the charge may become gradually red hot without clotting. The charge is thus left undisturbed for some time. During the first hour it is once or twice lightly rabbled; at the expiration of that time, after being well turned with the paddle, it will be found to be red hot throughout. The damper is then lowered so as to leave just sufficient draught for the free escape of the gaseous products of calcination. The working-doors are now left partially open to admit the air necessary for the oxidation of galena, and under its oxidizing action the charge soon acquires a high temperature. Care must, however, be taken to prevent the heat from increasing so much as to cause softening of the ore.

When the heat is properly managed, a white crust, consisting of a mixture of oxide and sulphate of lead, in which the former predominates, is rapidly formed on the surface of the charge, and no fumes are visible. The surface is renewed from time to time, about every quarter of an hour, either by rabbling or by paddling.

The requisite heat is maintained in the furnace during this process by firing with cinders only, which are preferable to coal for that purpose; not merely on account of their lower cost, but also because they give a steadier heat and do not yield gaseous hydrocarbons to interfere with oxidation.

The alternate raking and paddling of the charge is continued at regular intervals, until, on examination, it is thought to be sufficiently desulphurized, which is generally the case at the end of from four to four and a half hours.

The grate is then freed from clinker, coal is thrown on the fire, the damper is opened, and a brisk fire is got up; in a few minutes the heat in the furnace is thus raised so as to lead to a commencement of the period of reactions.

*b. Smelting.*—As soon as the firing has commenced, the ore lying at the back and extreme end of the furnace is raked towards the bridge. With the increase of temperature which now takes place, the roasted ore soon begins to soften and to give off white fumes, showing that the reactions which result in the liberation of metallic lead have commenced. Great care and attention on the part of the smelter are, however, necessary, in order to prevent too great a loss of lead by volatilization during the heating-up of the charge. The ore must on no account be allowed to liquefy, and as often as it shows a tendency to fuse, some slaked lime in powder is thrown on the charge, and well worked into it with a rake. By this means, and by carefully regulating the draught, the charge can be heated to the required temperature without fusion; lime must be frequently added during the process, but in small quantities at a time. The consumption of lime amounts, altogether, to about 2 per cent. of the ore treated.

Shortly after reaction commences globules of lead are seen on the surface of the charge, and before an hour has elapsed a certain quantity of lead has drained down the slope of the hearth into the well. The charge, having in due time attained the temperature of bright redness, must not be fired too hard. Both the draught and the firing are so regulated as to keep up the heat to the required degree without going beyond it.

The reactions are much aided by frequently rabbling and turning the ore; but as the working-doors must remain open for this purpose, it follows that a large quantity of air enters the furnace, oxidizing the sulphide in the ore, and so cooling the charge that the flow of lead becomes interrupted. When this cooling is judged to have been carried sufficiently far, the charge is rabbled, the doors closed, and the fire so urged as to fill the furnace with flame during several minutes. On re-opening the doors and paddling, the flow of lead recommences as before.

At the expiration of from two and a half to three hours from the commencement of the reactions, a considerable amount of lead will have accumulated in the well. A first tapping is then made, the lead being received in the tapping-pot, under which a small fire has been previously lighted in order to maintain the lead in a melted condition while it is being cast into pigs; the tap-hole is stopped with a plug of stiff clay.

The thick dross which rises to the surface of the lead, and which contains a certain quantity of sulphide removed from the charge by the hot lead, and again separated on cooling, is skimmed off with a shovel and put back into the furnace. A little fine coal, together with some burning cinders and lime, is now thrown on the lead, which is vigorously agitated with a small paddle and stirred until it is clean. It is then skimmed, the skimmings being put on one side, and the clean lead laded into moulds.

The firing, paddling, and cooling of the charge are repeated several times, until at last the residue becomes *dry* and gives out but little lead. Thereupon the heat of the furnace is considerably increased, but not in a sufficient degree to fuse or flow down the charge, and towards the close of the operation the matters remaining on the hearth consist, to a great extent, of oxides.

The pot-skimmings, composed of cinders and lead matte, are now thrown into the furnace and well paddled with the charge; the reaction of the sulphide of lead and cinders on the oxide and sulphate producing a further yield of lead.

When this has ceased, and it is seen that no more can be extracted without a very high heat and the addition of coal to the charge, the grey slag is raked out through the middle door at the back of the furnace, and the second and last tapping is made.

The whole period of reactions occupies from five to five and a half hours.

After the withdrawal of the slags, the bottom is examined, and if corroded into holes, or in any way injured, it is repaired by putting into the cavities a mixture of grey slag and lime, and beating it smooth with the paddle. It is of great importance to maintain the bottom perfectly even and with a good slope on all sides towards the tap-hole. A little lime is now spread over the bottom, and a fresh charge is at once let down into the furnace; the damper having been previously lowered to prevent loss of fine ore by the draught. The weight of coal consumed is equal to 40 per cent. of the ore smelted. The produce of lead per charge of 1,350 kilos. of ore at 81 per cent. is 15 pigs, weighing 901 kilos., and 290 kilos. of slags containing 50 per cent. of lead. Hence the total loss of lead in the reverberatory furnace is 3·52 units by volatilization, but of this a certain proportion is recovered from the fume collected in the condensers and flues. A larger percentage of lead in pigs might be obtained by adding more coal to the charge, towards the end of the process, and firing hard, in order to reduce part of the lead remaining in the slags. It is, however, considered more economical to limit the produce in the reverberatory to about 80 per cent. of the lead contained, and to carry the slags to the blast-furnace. This is especially the case when the blast-furnace is connected with good condensers and long flues.

It will be seen from the foregoing description that the process of reverberatory smelting adopted at Couëron differs principally from that generally employed in England, in there being no melting or flowing-down of the charge, as also in the long preliminary calcination of the ore before the reactions are commenced. The position of the tap-hole near the flue end of the furnace gives a longer slope to the hearth, and allows of heavier charges being worked; at the same time the lead produced is collected in the well at a greater distance from the fire.

Cornish Process.—This process was employed at Par, near St. Austell, where were situated the last lead-smelting works in operation in Cornwall. It was also formerly in use at Point, near Truro, as well as at the Falmouth Smelting Works; but smelting operations at all these establishments have been for some years abandoned. The lead ores treated by this method consisted principally of galena, usually containing a considerable amount of silver, with blende and various sulphurized ores of copper, associated with a siliceous gangue; their average yield of lead was between 60 and 70 per cent., and of silver about 35 ounces per ton.

This process comprehends two distinct operations, each conducted in a separate furnace:—

*a.* Calcination or roasting.
*b.* Flowing or smelting.

*a. Calcination.*—The furnaces used for this purpose varied considerably in size, some of them working charges of only 25 cwts., while others were capable of taking charges of 3 tons; the average charge may be taken at from 1 ton 18 cwts. to 2 tons. The Cornish calciner had generally three working-doors; two being in the longer sides and opposite one another, while the third was at the extremity of the longer axis of the hearth, in the immediate vicinity of the flue leading to the chimney. The widest part of the hearth was that between the two lateral doors; at the fire-place it had little more than one-third its maximum width, and at the opposite extremity its dimensions were gradually contracted to the width of the working-door. Immediately within the lateral doors were rectangular holes in the bed, through which the calcined ore was raked into an arched vault beneath. A passage through the fire-bridge, which enclosed a strong supporting bridge-plate of cast-iron, admitted of the free circulation of air; the charging was effected through a hole in the roof, usually covered by a fire-tile. This furnace was built of ordinary rubble-work, but was internally provided with a nine-inch lining of fire-brick. The working-doors had either strong cast-iron frames, or their sides were protected by blocks of granite. This calciner was worked by two men on each shift. Calcination was effected at a high temperature, and the charge was turned over once every hour; the operation was continued during from fifteen to eighteen hours, and a little lime was added from time to time to prevent clotting. About 6 cwts. of coal were consumed per ton of ore calcined.

*b. Flowing.*—The furnace employed for this operation was very similar to that used for the Flintshire process; the length of the hearth was about 14 feet, and its width 8 feet. This furnace had five working-doors, two on either side, and one in the end opposite the fire-bridge. The bottom, like that of the Couëron furnace, was supported on iron bearers, and finished with slags.

The charge, which consisted of 2 tons of calcined ore, was thrown in through the back doors, and was spread evenly over the sloping bed; the doors were then closed, luted with clay, and the heat raised. At the expiration of from two to three hours the charge had run down into the well, and, when rich ores were operated on, a first tapping was made at this period. The fused material was then dried up by being mixed with lime and culm, and again spread over the upper portions of the bed. About 2 cwts. of scrap-iron were now thrown into that part of the well nearest the tapping-hole, and a little fluor-spar was sometimes scattered over the surface of the charge. This done, the doors were again closed and luted, the charge re-melted, and the furnace subsequently tapped. This was done with a pointed iron bar, and the lead, which first flowed into the pot, was followed by a regulus chiefly derived from the action of scrap-iron upon galena. This regulus, *slurry*, sometimes together with a little of the lead, flowed over the lip of the tapping-pot into a small pit in the floor, and as soon as the slag began to make its appearance, the tapping-hole was enlarged and the lip of the pot was closed with ashes. The slag now flowed out upon the surface of the metal in the pot, which was almost surrounded, at top, by a strong iron hoop, and the molten slag was thus directed into a gutter, through which it flowed into a pit outside the building.

From the time of charging to the final tapping of the slag the operation occupied about eight hours; two men were employed in each shift, assisted by an attendant during the day shift. The consumption of coals was from 8 to 9 cwts. per ton of calcined ore treated; and the slags, which usually did not afford more than from one-half to one per cent. of lead by dry assay, were sufficiently poor to admit of being thrown away.

The principal portion of the copper contained in the ore became concentrated in the slurry, which was re-melted in order to extract the lead and silver it contained, and the resulting matte was then sufficiently rich in copper to be sold to the copper-smelters. The lead produced by this process was usually hard, and required to be softened by a process to be hereafter described. When, however, two tappings were made during the course of the operation, the lead obtained from that which took place after the first running-down of the charge was much softer, and more argentiferous, than that resulting from the final tapping.

The flowing furnace is sometimes used in North Wales and in various other localities, and is constructed and worked essentially in the same way as it was in Cornwall. It is also employed, instead of the slag-hearth, for the reduction of the rich grey slags from the Flintshire furnace by the aid of scrap-iron and carbonaceous matter, and is likewise used for smelting rich silver ores and jewellers' sweep.

In this process a large proportion of the lead is obtained by the mutual decomposition of galena and of the oxidized products resulting from calcination, the remainder being due to the reducing action of the culm and to the decomposition of sulphide of lead by metallic iron. It may therefore be regarded as a mixed process, comprehending the whole of the various reactions involved in the metallurgical treatment of lead ores.

SPANISH FURNACE, OR BOLICHE.—According to Pérès de Vargas, who lived in Spain during the reign of Philip II. (1556–1598), and who wrote on the metallurgy of that date, the *boliche* had been known from time immemorial, having been used by the Romans when in occupation of that country. Alonzo Barba, the celebrated priest of Potosi, states, on the contrary, that it was an Indian invention, introduced from America about the year 1640. M. Petitgand, the author of an excellent monograph entitled 'Exploitation et traitement des Plombs dans le Midi de l'Espagne,'[1] infers from this that it was first introduced from Spain into America, and subsequently re-imported into Spain, where its use and method of construction had become forgotten. However this may have been, there can be no doubt that the boliche is an exceedingly ancient furnace, and that for a very long period it was almost exclusively employed in the south of Spain for the reduction of lead ores.

It is constructed of rubble-work cemented together with a mortar formed either of clay or of vegetable soil, and is generally without any support from iron bars or binders. In place of these it is provided with strong buttresses at the angles, which are frequently carried upward, so as to form supports for the roof of the shed by which it is covered. The interior is protected by a lining of fire-brick, the hearth being composed either of *laja* or *laguena*, a description of clay resulting from the decomposition of talcose schists, or of a mixture of this clay with broken galena. The boliches employed by the various English companies established in the neighbourhood of Linares are constructed of sandstone, without any kind of lining, and are bound together by ordinary iron braces.

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[1] 'Revue Universelle des Mines,' Vol. ix. p. 297, 1861.

This furnace essentially consists of two arched chambers, one of which is employed for the reduction of the ore, while the only use of the second, which is situated between it and the chimney, seems to be to moderate the draught. These two chambers are separated by a fire-bridge, the second being generally in connection with a chimney about 30 feet in height. At Linares the boliches are worked in connection with the long condensing-flues of the establishment. The fire-place projects from one side of the hearth, is without a grate, and is 2 feet 2 inches in width and 5 feet 6 inches in length; the fuel, which consists of brushwood, such as cistus, broom, lavender, rosemary, juniper, &c., is supplied through a door at the end of the fire-place. The smelting hearth inclines towards the only working-door, at the extremity of its longer axis, and immediately within which is a receptacle in the floor for the collection of the molten metal. This door, and that through which the fuel is introduced, are the only openings in the furnace, with the exception of a smaller one supplying air to the fuel, and through which the ashes are withdrawn. The hearth, which is about 7 feet 6 inches in length by 6 feet 6 inches in width, has its angles rounded, and is sometimes made nearly circular in form. The second chamber, serving to equalize the draught, is lenticular in shape; its length, transversely to the chimney, is 12 feet, and its width 4 feet. At the extremity of its longer axis are doors, which are usually bricked up, but which are from time to time opened for the removal of fume and dust.

According to Mr. A. Dick, the smelting of a charge of lead ore in the boliche in the Linares district occupies eight hours, and comprehends three operations:—

*a.* *Caldeo*, or calcination, lasting from one to one and a half hour.

*b.* *Blandeo*, or *sweating*, occupies from four and a half to five hours.

*c.* *Corrida*, or *running down*, which occupies the remainder of an eight-hours' shift.

M. Petitgand, however, says that under favourable circumstances a *liga* or charge may be smelted in from four and a half to five hours. Mr. J. L. Thomas, among other details furnished to Dr. Percy, stated that a charge of ore, weighing 60 arrobas, or 13 cwts. 1 qr. 16 lbs. English, and yielding by assay 77·5 per cent. of lead, can be worked in from five and a half to six hours. It is evident, however, that the time necessary for working a charge will be much influenced by the richness and fusibility of the ores.

The ore, which is thrown in through the working-door by a scoop with a handle on either side, is spread evenly over the bottom and is stirred from time to time. When the operation of calcining is sufficiently advanced, the temperature is increased, and the running down of the charge commences. For drying up the slags, the ash and breeze falling into the ash-pit are alone employed. At the close of the operation the lead is tapped into a vessel, where it is first stirred up with dry leaves, and from which it is subsequently laded into moulds.

The yield of the boliche is about 80 per cent. of the contents of the ore by assay; the grey slags, which are drawn in the usual way, contain from 45 to 50 per cent. of lead, and amount to from 15 to 17 per cent. of the ore charged. They are reduced in a blast-furnace blown by bellows worked either by treadles or by horse-power. For smelting a charge of 60 arrobas of ore, from 1,550 to 1,750 lbs. (avoirdupois) of brushwood are required.

From having been subjected to a less elevated temperature, the lead smelted in the boliche is said to be softer than that obtained from the Flintshire furnace, but the difference in quality is by no means marked. The reactions which take place in this apparatus are of precisely the same character as those which occur in the furnace employed in North Wales, and the process is essentially one of double decompositions. At Linares, boliches have been constructed with fire-places adapted for the consumption of mineral fuel; but thus modified it appears to offer no advantages over the English reverberatory furnace; the amount of ore worked in a given time is smaller, and the proportionate consumption of fuel somewhat larger.

The boliche, although an ingenious contrivance for the reduction of rich lead ores by means of brushwood, is much inferior to the ordinary reverberatory furnace, when coal is to be obtained at a moderate price.

BLEIBERG PROCESS.—The galena treated at Bleiberg in Carinthia is accompanied by blende, calamine, willemite, cerussite, anglesite, and molybdate of lead. The ores of zinc, removed as completely as practicable by a careful system of mechanical treatment, are sold to zinc-works for reduction; while the lead ores are delivered to the smelter, either in a somewhat rough state of division, containing from 65 to 70 per cent. of lead, or as slimes, of which the yield is usually 5 per cent. less.

The furnace employed at Bleiberg differs essentially from those used in this country, since the hearth, instead of being broad, with a well in the middle and a tap-hole at the side, is long, and has a regular slope towards the door, which is placed at the extremity of its longer axis. The working-bed, about 10 feet long, and 4 feet 10 inches in greatest width, is rectangular for nearly half its length, and then contracts towards the working-door, which is 1 foot square. Besides having a regular slope towards this door, the hearth is hollowed from the two longer sides towards the middle, so that the metal may flow from every part of the bed into a receptacle in front of the working-door. The fuel commonly employed is wood, and the fire-place is parallel to the longer axis of the bed; the grate, which has a still greater inclination than the bed, is of stone, traversed by openings for the admission of the air necessary for combustion. These furnaces, which are constructed in pairs, are mainly built of red sandstone, and the bottom, which is 6 inches in thickness, is made of a mixture of clay, old bottoms, poor slimes, and slags fritted together by heat.

Brown coal is sometimes used as fuel in place of wood, and in such cases a grate consisting of iron bars is substituted for one of stone. Furnaces with two beds placed one above the other were for some time employed at the Imperial Lead-Works, at Bleiberg, but although the consumption of fuel was thereby reduced, the frequency of the repairs required and the uncertainty of the results obtained caused them to be abandoned.

The process of smelting is conducted as follows:—As soon as the hearth has become heated to dull redness the charge, weighing about 375 lbs. avoirdupois, is thrown in through the door, and, by a rake, is spread over every part of the bottom. The temperature is now kept so low that calcination may be effected without softening or agglomeration, and the ore is at short intervals turned over with a rake, and fresh surfaces exposed to oxidizing influences. At the expiration of from three to three and a half hours the roasting is generally found sufficiently advanced, and the temperature is then so raised as to determine the usual reactions between the unchanged galena and the oxidized products resulting from calcination. The rabble is now assiduously worked for the purpose of effecting a complete mixture of the charge, thus facilitating the production of metallic lead through the mutual decomposition of its sulphide and the oxidized products of roasting. This operation lasts from three and a half to four hours, and the lead, which is constantly liberated, flows down the inclined hollowed surface of the bottom into the receptacle placed in front of the working-door for that purpose. This lead, from its supposed purity, has received the name of "virgin lead," *Jungfernblei*, and in order to indicate its source of production was formerly sold in the form of irregular lumps as it fell from the furnace. At the expiration of the period above stated, from 125 to 150 lbs. of lead will have been collected, sulphide of lead will have almost entirely disappeared, and the residual grey slag will contain lead in an oxidized state only.

This slag is now dried up by the addition of ashes and breeze from the ash-pit, and the mixture, after being well worked with the rabble, is withdrawn from the furnace and laid aside to be treated with that resulting from the next operation. Another charge is introduced, and the process is conducted as before.

At the completion of the second stage the grey slag remaining on the bottom of the furnace is in its turn mixed with breeze, while that resulting from the working of the previous charge is added through the working-door; the furnace thus containing the slags resulting from the treatment of two distinct charges of ore. The temperature is now raised, small charcoal is added when necessary, and the whole intimately mixed with the rabble, in order to effect the reduction of the various oxidized compounds of lead. This reduction of the slag occupies from seven to eight hours, and results in the production of from 150 to 220 lbs. of lead; it consequently follows, that for the complete treatment of two charges, weighing together about 750 lbs. avoirdupois, from twenty-one to twenty-three hours are required. The yield of lead is usually, when the ores are of average richness, about 2½ per cent. less than that indicated by assay; the consumption of wood is about 11 cubic feet per Austrian centner (133·460 lbs. avoirdupois) of lead produced.

The workmen are paid at the rate of 1·80 florin per centner of lead extracted, and are debited 6·40 florins per klafter for the wood consumed. They also receive a bonus of 0·07 florin on each pound of lead produced in excess of the prescribed quantity, and are fined the same amount for every pound less than the amount allowed.

According to M. Landrin, the following allowances between the assay-results and furnace-produce were in force in 1857:—

| | | | | | |
|---|---|---|---|---|---|
| Assay result | 82 | per cent. : | tolerated loss | 2 | per cent. |
| „ | 80 | „ | „ | 3 | „ |
| „ | 78 | „ | „ | 4 | „ |
| „ | 76 | „ | „ | 5 | „ |
| „ | 74 | „ | „ | 6 | „ |
| „ | 72 | „ | „ | 7 | „ |
| „ | 70 | „ | „ | 8 | „ |
| „ | 68 | „ | „ | 9 | „ |
| „ | 66 | „ | „ | 10 | „ |
| „ | 64 | „ | „ | 11 | „ |
| „ | 62 | „ | „ | 12 | „ |
| „ | 60 | „ | „ | 13 | „ |

The charge of the Bleiberg furnaces has now been increased to about 465 lbs. avoirdupois.

REDUCTION BY METALLIC IRON IN REVERBERATORY FURNACES.—Silicate of lead, as well as the sulphurized and oxidized compounds of that metal, is reduced when strongly heated in contact with metallic iron; this operation is usually conducted in a low blast-furnace, but a reverberatory furnace has also been employed for the purpose.

The treatment of somewhat siliceous ores by fusion in a reverberatory furnace with either iron scrap or cast-iron was formerly carried on to a limited extent in France, but was ultimately discontinued, on account of the great cost, and the unsatisfactory nature of the results. The furnace employed sloped from the fire-place to the chimney, placed at the opposite extremity of the hearth, where there was a working-door with a tapping-hole beneath it, in front of which was the usual reservoir for the reduced metal. The charge consisted of about 400 kilos. of galena, containing nearly 80 per cent. of lead, mixed with from 100 to 120 kilos. of scrap or of cast-iron. When the temperature had become sufficiently elevated the charge was stirred at frequent intervals, and lead became reduced at the expense of the iron, which was converted into sulphide.

From galena containing 80 per cent. of lead 67 to 70 per cent. of metal was extracted, the matte containing from 5 to 12 per cent. of lead; 4 per cent. passed off by volatilization, and the slags retained from 4 to 5 per cent.

The entire contents of the furnace were run off into the tapping-pot which retained the lead, while the principal portion of the matte and slag flowed over upon the floor. The last two products may be re-treated in a blast-furnace for the lead they contain, but the process of smelting raw ores with iron in reverberatory furnaces is wasteful and expensive.

The processes employed in different localities for the reduction of lead ores in reverberatory furnaces are exceedingly various, their adoption being determined by the nature of the ores, the quality and price of fuel, and the nature of the fluxes available. The methods described may, however, be considered as typical of those employed in the principal lead-producing centres, although they are sometimes more or less modified to suit the exigencies of local circumstances.

## SMELTING IN BLAST-FURNACES.

**SLAG-HEARTH.**—The various rich slags resulting from the different operations of a lead-smelting establishment are either treated in the furnaces in which they are produced, as in the case of the Cornish and Bleiberg processes, or they are reduced in a blast-furnace specially employed for that purpose. In this country the slag-hearth is still used, although its application is no longer so universal as formerly.

This is a deep hearth, or shallow rectangular furnace, blown by one tuyer, made of fire-brick and cast-iron, cased, with the exception of the front, in an exterior covering of ordinary brickwork, and bound by iron braces. The casing is extended upwards so as to form a chimney, which at the top is connected with the system of flues for the condensation of lead fume with which every well-appointed establishment is provided. The depth of this furnace from the front to the tuyer is about 30 inches, its width may be 22 inches, and its height a little more than 3 feet. The bottom is composed of a thick cast-iron *bed-plate*, laid with a slight inclination from the back towards the front of the furnace; on the bed-plate are placed cast-iron *bearers*, which carry the side walls of the furnace. The front is partially closed by a plate of cast-iron, called the *fore-stone*, and the back below the tuyer is formed of another thick plate of the same metal. A space of from 5 to 7 inches is left between the bottom of the fore-stone and the cast-iron plate forming the bottom. In front stands the *lead-trough*, divided into two unequal portions by a division forming part of the casting; the larger one corresponds, in width, with the bed-plate under which its edge is introduced, and at the bottom of the partition is a hole through which the molten metal flows into the smaller division of the trough. The slags, escaping from this furnace through an opening in the stopping of the breast, pass over the edge of the larger division of the lead-pan and then flow into a cistern sunk in the ground, through which flows a stream of water. This determines the disintegration of the slag, and allows of the separation of any enclosed metallic shot. Before beginning to work a furnace of this description, the bed-plate is covered with a layer of well-burnt and coarsely sifted ashes to within about an inch below the level of the tuyer; and the ash bottom thus formed is made to slope in the same direction as the plate itself. The space between the bearers in front of the ash bottom is now closed with clay, &c., up to the lower edge of the fore-stone, and in this stopping an opening is made for the escape of the slag. The bottom of spongy cinders, which is somewhat closely beaten together, acts as a filter, by which the metal is separated from the slags which flow over its surface, and escape by the opening in the breast; while the heavier lead falls to the bottom of the hearth and, percolating through the ash bottom, is conducted by the bed-plate into the lead-pot. The larger division of this pot is filled with cinders, which, as in the former case, act as a filter, by means of which the molten lead is separated from the less dense and more viscous slag. The fuel employed is coke with a little peat, and the products are *slag-lead* and a vitreous *black slag*, sufficiently poor to be thrown away.

Blocks of peat are first piled upon the bottom of the furnace, and one of these is ignited and thrown before the tuyer. When the peat has thus become fairly ignited, coke is thrown in; and as soon as this appears to be properly lighted, grey slag, with some *browse* (partially reduced slags removed from the hearth at the termination of a previous operation) and a little black slag are introduced. From this time the hearth is supplied with alternate charges of fuel and slag, and when a sufficient quantity of the latter has been melted, which happens shortly after the furnace has been set in operation, the smelter, with an iron bar, makes a hole in the centre of the stopping between the bed and the lower edge of the fore-stone. Through this the melted slags make their escape, and the furnace being now in its normal working condition, the several operations are repeated, throughout the shift, without variation.

At the close of each shift, which often lasts about eight hours (six of which will be employed in smelting and the other two in cleaning and preparing the hearth for the next working period of similar duration), the blast is continued for about half an hour without any further addition of either fuel or slag.

As soon as the blast has been cut off, the clay stopping beneath the fore-stone is removed, and the ash-bed, with as much as possible of the adhering slaggy matter, is taken out; this, under the name of *slag-hearth browse*, is reserved for treatment during the course of the next shift. Finally a few bucketfuls of water are thrown into the hearth, in order to put out the fire and to cool it previous to the clearing or *stubbing-out* necessary before commencing another shift.

The hearth is worked by two men—a smelter and his assistant. The duration of a shift varies in different localities from eight to sixteen hours, and the production of lead and consumption of fuel are greater or less in accordance with the nature and richness of the materials treated. According to H. L. Pattinson, the slag-hearth shift in the North of

England (1832) lasted from fourteen to sixteen hours, and produced from 10 to 21 cwts. of lead; from 15 to 18 cwts. of coke being required to produce 21 cwts. of lead. From the nature of the material from which it is obtained, and the high temperature employed for its production, slag-lead is invariably hard, and requires softening before it can be applied to the purposes for which ordinary lead is used; it is laded out from the smaller division of the lead-pan and cast into pigs. When grey slags are found to work with difficulty in the slag-hearth, the addition of tap-cinder is often found advantageous. The mixture smelted in the slag-hearth should not contain above 35 per cent. of lead, and when richer than this should be mixed with black slag or some other poor material.

The Spanish slag-hearth, which has been adopted at some of the lead-works in this country, is circular, and blown by three tuyers. Its general construction very closely resembles that of the Castilian furnace, but it is without iron columns, and is connected with the main flue by a chimney. It is worked in the same way as the Castilian furnace, but the slags are sometimes run into water.

CASTILIAN FURNACE.—This furnace, which is circular, is usually about 3 feet internal diameter, and is constructed of fire-bricks, so moulded as to fit together and allow all the various joints to follow the radii of the circle described by the brickwork. Its height is about 8 feet, and the thickness of the masonry is usually 9 inches. In this arrangement the breast is formed by a deep semicircular iron pan, furnished with a lip for running off the slags, and a longitudinal slot for the convenience of tapping. On the top of the brickwork a box-shaped covering of masonry is supported by a cast-iron framing resting on four pillars, and in this is placed a feeding-door, and the outlet by which the products of combustion, &c., escape into the flues. The lower part of this hood is fitted to the body of the furnace, whilst its top is closed by an arch of brickwork laid in clay. The bottom consists of a mixture of fire-clay and coke-dust slightly moistened and well beaten in to the height of the top of the breast-pan, which may be about 3 feet above the floor level. Above the breast-pan an arch is frequently so turned that, when the breast has been built up, it may form a niche 9 inches in width and 12 inches in height. When the bottom has been properly beaten to the required height, it is so hollowed out as to form an internal cavity communicating freely with the breast-pan, which is likewise filled with brasque, and subsequently hollowed to the depth of the internal basin of the furnace. The blast is applied by three tuyers, each 3 inches in diameter at the smaller end, and 5½ inches at the other, into which the nozzles are introduced. The blast is generally produced by a fan, and is conveyed to the tuyers through brick channels beneath the floor of the furnace-shed.

The mixture of plumbiferous matter treated in this apparatus ought never to contain above 30 per cent. of lead, and if richer, should be reduced to this tenure by the addition of a sufficient amount of poor slags. In charging, the coke should be thrown to the middle, while the matters to be treated are spread around next the brickwork. In this way the furnace is prevented from becoming too hot, and the bricks are consequently preserved for a longer period than they would otherwise be.

For the purpose of allowing the slags to escape into the breast-pan, a hole is made in front at the height of the fore-hearth, which, to prevent the cooling of the slag, is kept covered by a layer of coke-dust or cinders. From the breast-pan the slags flow continuously through an iron spout into cast-iron waggons, where they consolidate into masses, having the form of inverted truncated pyramids, of which the larger base is about 20 inches square. When a sufficient amount of lead has accumulated in the bottom of the furnace, it is tapped into an iron pot by removing a plug of clay from the tap-hole situated in a slot in the breast-plate, and, after being skimmed, is laded into moulds.

The waggons in which the liquid slag is received run either on iron plates or on a tramway, by which, when one is removed, its place is supplied by another. When cold, the bodies of the waggons are turned over and the blocks of slag removed. One of the advantages obtained by this method of working arises from the circumstance that, should the furnace at any time run lead or matte, without its being observed by the smelter, the whole of it will collect at the bottom of the waggon, where the block is contracted, and from which any metal or regulus may be removed when the mass has sufficiently cooled.

These furnaces will smelt rich slags and other plumbiferous matters with an expenditure of about 10 per cent. of coke, while the slags obtained from them should not contain above 1 per cent. of lead.

In working this furnace a little scrap iron is generally used, and care should be taken to prevent flame from appearing at its mouth; since, provided the slags are liquid and flow readily off, the cooler the furnace can be kept at top the less will be the loss of lead through volatilization. In addition to great attention being paid to the working of the furnace, it is necessary that the establishment should be provided with extensive flues, in which condensation of the fume may take place before arriving at the chimney through which the more volatile matters escape. As an instance of the economical working of these furnaces, it may be stated that slags giving, by fire assay, only 8 per cent. of lead, with traces of silver of no commercial value, have been treated with advantage. In Derbyshire, where large heaps of slag of the above percentage were worked, some years ago, in the Castilian furnace, only 3½ per cent. was directly obtained in the metallic form, while 4 per cent. was recovered from the flues in the state of fume, and subsequently reduced in a reverberatory furnace.

When ores of lead are treated in the blast-furnace, they are usually either smelted in the raw state with metallic iron, or are first roasted and subsequently fused with iron or a ferruginous flux.

**Smelting Raw Ores with Metallic Iron.**—This process was at one time extensively employed on the continent of Europe for the reduction of siliceous lead ores. In Silesia, where it was long used, it has been replaced by reverberatory smelting, while in the Upper Harz it has been superseded by a process in which basic silicate of iron, in the form of finery slags, has been substituted for metallic iron.

The reduction of lead ores, consisting principally of galena associated with antimonial ores, quartz, and spathic iron ore, was formerly conducted at Clausthal as follows:—

The mixture treated consisted of 34 centners of picked ore and slime, containing about 56 per cent. lead and 36 ounces of silver per ton; 4 to 5 centners of *Heerd* or silver-refinery bottoms impregnated with litharge; 1 centner of *Abstrich*, or antimonial litharge, from the refinery; 39 centners of slags, derived either from the first fusion of the ore treated, or from the re-melting of mattes; and 4½ centners of granulated cast-iron.

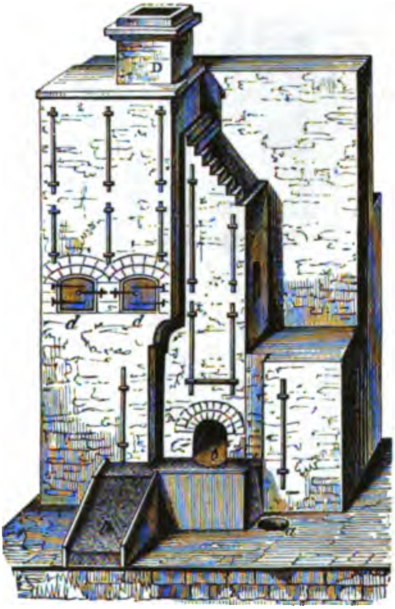

Fig. 184.—Old Furnace, Clausthal; front view.

The fusion of this mixture was conducted, with soft wood charcoal as fuel, in a small blast-furnace of from 20 to 25 feet in height, and about 3 feet square in the widest part. The hearth of the furnace was so arranged as to extend beyond the breast into a small raised platform situated immediately before it. The lining of the hearth was of refractory fire-stone, and the bottom was hollowed out of brasque in such a way as to afford a gentle slope from the tuyer to beyond the front wall. A tapping-hole entered the lowest part of this basin, and afforded a means of drawing off its contents when accumulated in sufficient quantity. This receiving basin, *a*, fig. 184, was on a level with the floor and at some distance from the breast of the furnace, which was supplied with a blast through a tuyer situated at *t*, fig. 185, which represents a vertical section of the same furnace.

In charging the mineral, care must be taken to direct it towards the side of the tuyer, while the fuel is chiefly thrown towards the breast. The cold air constantly entering through the tuyer cools the slag produced in its immediate vicinity, and forms around the nozzle a circular prolongation of six or seven inches in length, on the proper management of which depends, in a great measure, the success of the operation. One of the principal uses of these nozzles is to prevent oxidation of lead; since the blast being led to the fuel in the centre of the hearth through, but out of immediate contact with, the mineral charged at the back of the furnace, does not so readily give rise to the formation of fusible silicates of lead.

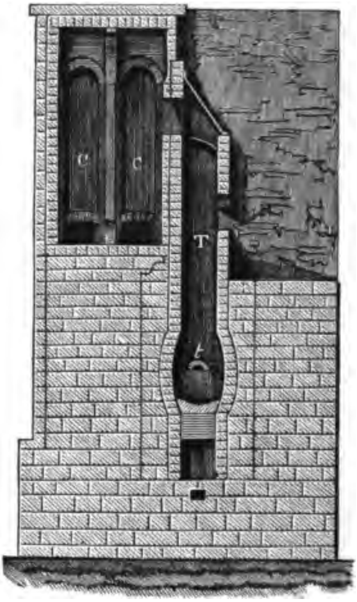

Fig. 185.—Old Furnace, Clausthal; vertical section.

With this view the smelter should attend to the proper regulation of the length of the slag-nozzle, as by it the economical working of the furnace is materially affected. It is also necessary, by regulating the supply of air and fuel, to so arrange the temperature that the upper part of the shaft may not become too strongly heated, as in that case considerable quantities of lead are driven off. With all these precautions, there is, however, a considerable loss from sublimation, and therefore the gases passing from the throat, T, are passed through chambers, C, fig. 185, before escaping into the atmosphere by the chimney, D.

In these chambers large quantities of fume accumulate, which is occasionally removed through the doors, *d*, fig. 184, for the purpose of being mixed with other lead products, and again treated in the furnace.

During the whole time this furnace is in action the slags flow continuously into the fore-hearth, where, being solidified, they are seized with a stout iron hook, and dragged down the inclined plane, *p*, to the foundry floor. When the basin, *b*, has become filled with metallic lead, the stopping is removed from the tapping-hole communicating with the reservoir, *a*, into which the metal is drawn off.

The products thus run off into the outer basin divide into two parts. The lower portion is metallic lead, while the higher consists of sulphide of lead, more or less mixed with sulphides of other metals originally present in the ore, and particularly with sulphide of iron, resulting from the decomposition of galena by that metal. This substance, which readily solidifies, is the *first matte* (or *schliegbleistein*), and is removed from the surface of the bath by an iron eye, around which a circular cake is allowed to solidify, and is stowed away for subsequent treatment. The lead is afterwards laded into moulds. The poorer slags are thrown away, while those which contain lead are added as flux in a future operation. When a sufficient quantity of rich slag is not to be procured, some of the poorer slags are used for this purpose; but this never takes place excepting when a proper supply of the richer kind is not to be obtained. The products resulting from a mixture having the composition before given, consisted of 19 centners of lead and 8 centners of first matte, containing from 30 to 35 per cent. of lead.

When a considerable quantity of these first mattes had accumulated in the establishment they were roasted in heaps, laid on a stratum of fuel, and by this means large quantities of sulphur and sulphurous anhydride were disengaged.

This first roasting occupied from three to four weeks, at the expiration of which time the heap was carefully picked over and the products divided into two classes; those portions which had been sufficiently roasted being again taken to the furnace and re-treated, while fragments which still retained a considerable amount of sulphur were subjected to a second process of roasting. By this process eight to nine successive roastings were necessary before the whole of the matte was obtained in a fit state for smelting.

When sufficiently roasted, the mattes were fused in a small blast-furnace, after being mixed in about the following proportions with various other materials:—

| | | |
|---|---|---|
| 32 | centners | of roasted matte. |
| 30 | ,, | of rich ore-furnace slags requiring cleaning. |
| 4 to 5 | ,, | of Heerd. |
| 2 | ,, | of Abstrich. |
| 2 | ,, | of slags from the litharge-reviving furnace. |
| 1 | ,, | of granulated cast-iron. |

The furnace in which this mixture was introduced was about 5 feet in height, and considerably contracted in the vicinity of the hearth, which, as in the case of the larger apparatus, was provided with a sloping fore-hearth, and a distinct tapping-basin for the reception of the metallic products.

The fuel employed was mixed charcoal and coke, and the blast, supplied by a single tuyer, was conducted into the furnace through a slag-nozzle about 3 inches in length. During the process of roasting, the larger portion of the sulphide of iron passed to the state of peroxide, and during the subsequent fusion, this oxide, partially reduced by the fuel, became protoxide, and united with the siliceous matters present to form a vitreous fusible slag; which, flowing through the aperture of the fore-hearth, was continually removed. The sulphide of lead was at the same time reduced through the agency of metallic iron, and new mattes somewhat analogous in their composition to the first were obtained.

These, when sufficiently solidified by cooling, were removed in the way already described, after which the lead was taken out in iron ladles and cast into pigs.

The treatment of 32 centners of roasted matte, with its associated fluxes and other products, afforded, at Clausthal, 12 centners of metallic lead and 8 centners of *second matte.*

The second mattes were subjected to a treatment similar to that employed for those obtained by the direct treatment of the ores. They first underwent one or more roastings, and were subsequently treated in the same furnace and with the same additions as those employed in the case of first mattes. In this way a further amount of metallic lead and a *third matte* were obtained; this was again roasted, fused with a proper addition of fluxes and other matters, and metallic lead and a *fourth matte* were the result. The copper, of which a small quantity only is contained in the original ore, having a greater affinity for sulphur than is possessed by lead, accumulated in these mattes, which, after the fourth roasting and fusion, became rich in that metal. The sulphide last obtained, known by the name of *copper matte*, or *Kupferstein*, was subsequently treated for coarse copper.

The above process has been so far modified in late years that, while still retaining the method of desulphurizing the ore in the furnace, it is no longer done by metallic iron, but by slags containing from 60 to 70 per cent. of ferrous oxide, brought from the Lower Harz copper-works. The lead matte, or *Bleistein*, is now first burnt in kilns, finished in open heaps, and constantly returned to the ore-smelting furnace until the contained copper is concentrated to about 13 per cent., when it is passed on to a furnace for the production of copper regulus. The furnaces now used at Clausthal are circular in form, with a working height of about 16½ feet above the tuyers, 6½ feet in diameter at the throat, and 3 feet in the hearth. A tube 20 inches in diameter plunges about 2½ feet into the top of the column of materials, and carries off the gases to the flue-chambers. Blast is supplied through four water-tuyers, and the consumption of fuel is 40·6 of coke and 2·5 of charcoal per 100 parts of raw ore smelted. From 7½ to 8 tons of materials are passed through the furnace in twenty-four hours. The slags are re-smelted when containing 5 per cent. of lead, and about 6 dwts. of silver per ton.

TREATMENT OF LEAD ORES BY ROASTING AND SUBSEQUENT SMELTING WITH METALLIC IRON.—The method of smelting employed at Pontgibaud[1] affords a good example of the treatment of highly siliceous lead ores by roasting and subsequent fusion with metallic iron in the blast-furnace. The average produce of the stuff, as extracted from the mines, is about 5 per cent., and consequently large quantities have to pass through the various processes of crushing and washing in order to obtain the amount of ore, containing on an average 50 per cent. of lead, annually treated in the establishment. As much cobbed ore as practicable is, however, selected by hand picking, in order to avoid, as far as possible, the loss incident to mechanical preparation. The ores, which vary considerably in richness, both as regards lead and silver, are delivered to the works in a ground state; the gangue is always highly siliceous, but blende, barium sulphate, arsenical and ordinary iron pyrites are also present.

All assays are made in iron crucibles, and, when properly conducted, yield results nearly as high as those obtained by the wet way; this is doubtless owing to the impurities in the button of lead compensating for loss by volatilization.

The process employed at Pontgibaud for the treatment of argentiferous galena comprehends two distinct operations, namely:—

*a.* Calcining and fluxing in a reverberatory furnace.

*b.* Smelting the fluxed ore with metallic iron in a blast-furnace.

*a. Calcining and fluxing.*—Although the ores do not materially differ as regards the nature of their gangue, they vary considerably in richness, and consequently in the proportion of earthy matter they contain. It has therefore been found desirable, before commencing their treatment, to prepare a uniform mixture from all the parcels of ore available. On this greatly depends the regularity of the subsequent operations, and in a great measure also their economical working.

As it would be difficult to thoroughly mix the whole of the various parcels of ore, sometimes amounting to several hundred tons, a *lit de grillage* of 20 tons (20,000 kilos.), or a little more than the quantity usually calcined per diem, is prepared. This is done by weighing out and spreading in thin layers, one above another, the exact proportion of 20 tons which each parcel bears to the total quantity in stock. The bed being thus prepared, the charges are obtained by cutting down, perpendicularly, with a shovel, the pile of stratified ore, in such a way that every ton of the mixture removed shall have the composition of the entire mass. To this mixture of ores are added the calcined mattes resulting from smelting in the blast-furnace; these usually contain about 17 per cent. of lead and 14 ounces of silver per ton, and constitute about 10 per cent. of the charge. The object of adding them to the lits de grillage is, that the oxide of iron may serve as a flux at the close of the operation of roasting. As the ores are always very quartzose a little mill-cinder is added in addition to the calcined matte.

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[1] The mines and smelting works of Pontgibaud, Puy-de-Dôme, France, have, since 1853, been carried on under English management.

This mixture is calcined in large reverberatory furnaces, worked from both sides, 40 feet in length, outside measure, and 15 feet in width at the widest part. The exterior is built of cut lava, and the sides, bottom, and roof are of fire-brick. This furnace is provided with twelve working-doors, and is, as before stated, furnished with a brick bottom. Of the six doors on each side of the furnace, two correspond with each charge, and enable the workmen to turn or advance the ore when required.

The different parts of the furnace occupied in succession by every charge may be distinguished as follows:—

1st. Drying-bed immediately under the charging-hoppers and furthest removed from the fire-place.

2nd. Desulphurizing or oxidizing-bed, occupying the middle and widest part of the furnace.

3rd. Fluxing-bed, next the fire-place.

The first two are on the same level, and the third about 6 inches lower than the two others.

At intervals of six hours, a charge of fused ore is withdrawn by tapping, and the other charges in the furnace are advanced a stage in the direction of the fire-place, while a fresh charge is let down from the hoppers upon the drying-bed beneath. The time each charge remains in the furnace is consequently eighteen hours; 8 tons (8,000 kilos.) are thus roasted and fluxed in the course of twenty-four hours, with a consumption of about 2 tons (2,000 kilos.) of coal, and the addition of 6 per cent. of lime. The amount of iron slag added is about 7 per cent.; but whatever the proportion of slag or lime may be, the weight of ore charged remains constant. Four men per shift of twelve hours are employed at each furnace, and the loss of lead during the operation is from 2 to 3 per cent.

*b. Smelting.*—The fusion of the roasted ore and fluxes was for many years effected in Castilian furnaces constructed of blocks of cut lava, of which fig. 186 represents a front elevation. Their height from the slag-overflow, *a*, to the charging-door at the back, was 5 feet, and their internal diameter 35 inches; diameter of tuyers, 3 inches; pressure of blast, 4 inches of water. The blast was supplied by the nozzles, *b*, of which there were three in connection with the mains, *c*; the tapping-hole was in the opening, *d*. Water-tuyers were not used.

The *lits de fusion* were composed as follows:—

| | Kilos. |
|---|---|
| Roasted ore . . . . . . . | 10,000 |
| Scrap-iron . . . . . . . . | 1,000 |
| Limestone . . . . . . . . | 1,600 |
| Fluor-spar . . . . . . . . | 300 |

The above figures give the average quantities of flux in the furnace mixtures; they were, however, modified with the nature of the ores, the proportions of limestone and fluor-spar varying most considerably.

The mode of charging was similar to that employed for other furnaces of this kind, the ore being principally distributed around the sides, and the coke kept in the middle and towards the breast. This furnace was maintained constantly full, and care was taken not to allow flame to appear above the top of the charge; the breast-pan was large and held from 15 to 20 pigs of lead.

From 13 to 15 tons of roasted ore were smelted in the course of twenty-four hours, with a consumption of about 1 ton of coke; the quantity of lead obtained during the same time was from 100 to 120 pigs, or from 5 to 6 tons. In addition to lead, the ores yielded from 7 to 10 per cent. of matte.

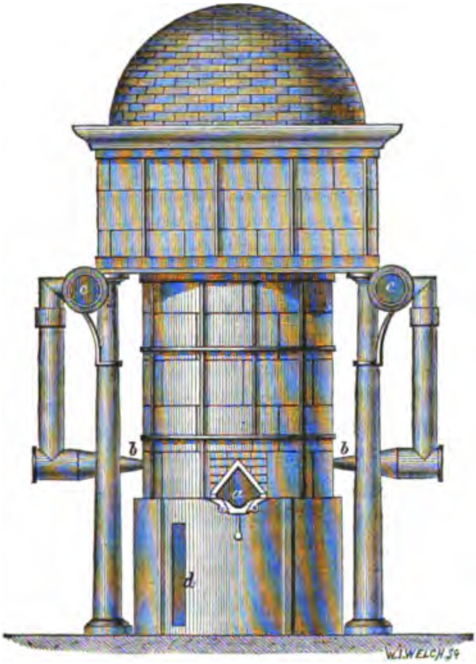

Fig. 186.—Castilian Furnace, Pontgibaud; elevation.

The slags, under the most favourable circumstances, contained nearly 2 per cent. of lead; when it exceeded 3 per cent. they were re-smelted. It was found, by assay, that the slags poorest in lead contained at least 40 per cent. of ferrous oxide, but that this might be partially replaced by lime, especially when fluor-spar was added to the charge. Slags in which oxide of iron had been replaced by lime were, however, never so poor in lead as those having the normal composition, while the amount of that metal volatilized was considerably increased. The proportion of slags produced was from 65 to 70 per cent. of the ore smelted, and as they flowed from the furnace they were received into cast-iron waggons, which when full were drawn to the waste heaps and tipped.

The lead obtained from the Castilian furnace contained almost the whole of the silver originally present in the ores, except the comparatively small proportion combined with the mattes or retained in the slags. The usual assay of the mattes was from 15 to 20 per cent. of lead and from 12 to 16 ounces of silver per ton; the amount of silver per ton (1,000 kilos.) of lead varied from about 95 to 125 ounces, since not only do the Pontgibaud ores vary in their yield of this metal, but other ores are sometimes purchased and smelted with them. The whole of the silver in the mattes, and a portion of that in the slags, was recovered during subsequent operations, but traces of that metal are nevertheless unavoidably lost. According to assay, this loss amounted to 6·12 dwts. per ton of slag, or about 0·568 per cent. of the total quantity contained in the ores. A certain amount of silver was also volatilized with the lead, but what proportion was lost from this cause could not be easily determined. That it was, however, very small is probable, from the known behaviour of silver in a reducing atmosphere, and from the low proportion of that metal found by assay in the fumes collected from the flues and condensers.

The proportion of silver thus carried off, and subsequently recovered in all the various processes, including cupellation, amounted to only 0·470 per cent. of the total quantity contained in the ores.

The loss of lead in smelting ores in the blast-furnace amounted to 17 per cent. of the quantity contained in them; 2 per cent. of this was subsequently recovered from slags by re-smelting, and about 3½ per cent. from fumes. The actual loss in the operation was consequently equal to 11¾ per cent. of the total quantity indicated by assay as contained in the ores.

The fume collected from the flues was subsequently mixed with siliceous ores, and fused in a calcining furnace. The fused mass thus obtained was smelted in the blast-furnace like ordinary ores. The fume and ore were mixed in the following proportions, and charged into the calcining furnace, through the working-doors, upon the middle bed.

Fume 60 parts, assay 62·7 per cent. lead.
Ore 40 „ „ 40·0 „ „

Each charge weighed 2½ tons, and five charges, or 12½ tons, could be passed through a furnace in twenty-four hours; the loss of weight experienced being about 15 per cent. The same number of men were employed as for roasting ore; the consumption of coal was 17 per cent., and of lime 3 per cent., of the weight of calcined stuff.

The furnace-mixtures, for fume and ore, were composed as follows:—

| | Kilos. |
|---|---|
| Fume and ore | 10,000 |
| Iron | 1,200 |
| Limestone | 3,500 |
| Fluor-spar | 300 |
| | 15,000 |

The above quantity was smelted in the Castilian furnace in twenty-four hours, with a consumption of 9 per cent. of coke.

The quantity of fume annually collected was about 150 tons, averaging 57 per cent. of lead and 4 ounces 5 dwts. of silver per ton. Its richness varied in different parts of the flue, but in general the percentage of lead increased, while the proportion of silver decreased with the distance from the furnaces. The lead recovered from fume amounted to 78·57 per cent. of the quantity contained in it, as indicated by assay; or 3·67 per cent. of the total quantity in the ore.

The lead produced from the blast-furnace required to be softened before desilverization by Pattinson's process. This was effected by exposing it at a low red-heat to partial oxidation in a reverberatory furnace.

*Losses of Lead and Silver.*—From 100 parts of lead contained in the ores treated, 85·75 parts were obtained, either directly or from fumes and slags, namely—

| | Per Cent. |
|---|---|
| From ores | 80·04 |
| From fumes | 3·67 |
| From slags | 2·04 |
| | 85·75 |

In desilverizing the lead by crystallization a loss of 3·25 per cent. was experienced, the total weight of poor lead produced for sale being 82·50 per cent. of the quantity contained in the ores. The loss in desilverizing was distributed as follows—

| | Per Cent. |
|---|---|
| Refining and reducing | 1·25 |
| Improving and reducing | 2·00 |
| | 3·25 |

The loss of lead in the three principal divisions of the Pontgibaud process was therefore—

| | Per Cent. |
|---|---|
| In roasting | 2·50 |
| In smelting | 11·75 |
| In desilverizing | 3·25 |
| | 17·50 |

The percentage losses on the total quantity of silver obtained from the ore were—

| | Per Cent. |
|---|---|
| In slags . . . . . . . . | 0·568 |
| In market lead . . . . . . . | 0·533 |
| | 1·101[1] |

By the processes employed when the Castilian furnace was in operation the annual production of silver is from 3½ to 4 per cent. in excess of that indicated by assay of the ores treated.

Of 100 parts of silver produced

98·82 were obtained direct from the ores,
0·64 ,, from the slags,
0·54 ,, from the fumes.

The quantities of lead and silver produced at the Pontgibaud smelting works (from local and purchased ores) for the year 1880 were as follow:—

| | |
|---|---|
| Lead . . . . . . . | 1,424 tons. |
| Silver . . . . . . . | 174,012 ounces. |

The Castilian furnace has now, for some time, been replaced at Pontgibaud by the water-jacket furnace, figs. 187, 188, described at pp. 618, 619, as being employed at Couëron. These new furnaces, in a given time, get through a larger amount of material than the old ones, with only a slightly increased expenditure of fuel. The action of the water-jacket furnace is continuous, so that there is no loss of time, and the expense of repairs is much reduced. The calcination and fluxing of the ores at these works previous to smelting are conducted as formerly, the furnace-mixtures being exactly similar to those used in the Castilian furnace—the general routine of the operations in the establishment remaining unchanged, except that the separation of silver from lead is now effected by zinc.

Treatment of Siliceous Ores at Couëron.—This treatment comprehends two distinct operations:[2]—

*a.* Calcination in the reverberatory furnace.

*b.* Reduction in the blast-furnace, with lime and basic silicates of iron as fluxes.

*a. Calcination.*—The reverberatory furnace employed for this purpose is similar in form and dimensions to that represented in figs. 181, 182, 183, p. 591, 592; the only difference in the two being that the furnace used for roasting has no tapping-pot, and has the bottom filled with black slag, level with the working doors, so as to form a flat hearth on which the charge may be uniformly spread and exposed to the fire. The ores roasted in this furnace contain from 50 to 60 per cent. of lead, with variable proportions of silica. They are worked in charges of 1,400 kilos., or about 28 cwts., each.

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[1] As indicated by the assay of slags and market lead; the actual production was slightly in excess of that found by the assay of ores.

[2] This method of treating lead ores was introduced at Couëron by the late Mr. William Hutchison.

Each charge is placed ready in the hopper above the furnace in order that it may be let down when required, without loss of time. Ores which are not sufficiently small are ground, and sifted through sieves with meshes $\frac{1}{8}$ of an inch in diameter.

The interior of the furnace being red hot, the charge is introduced and at once spread evenly over the hearth; the damper having been previously run down to the lowest point so as to reduce the draught, and thereby prevent mechanical loss of fine ore. The working-doors are now closed, and the charge is left undisturbed until decrepitation has entirely ceased and the ore has become partially red hot. It is then carefully rabbled, so as to expose the under parts to the heat, and a few minutes afterwards is turned with paddles. In about an hour the charge will have acquired a uniform dull-red heat. More air is then admitted by partially opening the fire-door and the two working-doors nearest the bridge. The damper is at the same time so adjusted as to afford only sufficient draught for the free escape of sulphurous fumes. The charge is turned or rabbled at intervals of about a quarter of an hour, so as to expose fresh surfaces to the oxidizing action of the air, and, by that means, to convert the sulphides into oxides and sulphates with evolution of sulphurous anhydride.

The partial opening of the doors for the admission of air has the effect of cooling the charge below the temperature requisite for oxidation, and in order to maintain it at the proper dull-red heat, it is necessary to close, from time to time, all the doors, and to throw a few shovelsful of fuel on the fire. The fuel preferred at this stage of the process is cinders, since in burning they yield but little flame; care must, however, be taken not to raise the heat beyond the proper point, or it will cause the ore to clot, thereby rendering its subsequent desulphurization, within a reasonable time, impossible. Whenever it evinces a tendency to soften, the doors are at once opened and the damper is raised, until the charge has cooled to the proper temperature.

The alternate heating and cooling of the charge is repeated at frequent intervals during the entire process, and in such a way that, while furnished with a plentiful supply of air, it is maintained, as nearly as possible, at a dull-red heat without clotting. The ore is raked and turned at regularly recurring intervals, until at the end of six hours it contains a sufficient amount of oxide and sulphate of lead to effect the decomposition of a large portion of the remaining sulphide of lead during the subsequent stage of agglomeration. Six hours are generally found to be sufficient to determine the necessary desulphurization of ordinary ores. At the expiration of that time the heat is considerably increased in order to effect the agglomeration of the charge and to give it the cohesion necessary for passing through the blast-furnace. The grate is therefore cleaned from clinker, the damper drawn up, and fresh coals put on until a good fire is obtained. Under the influence of the increasing temperature the charge soon begins to soften, and at the same time it is constantly raked, turned with the paddle, and advanced towards the bridge. In proportion as it melts it is raked through one of the working-doors upon the floor of the furnace-

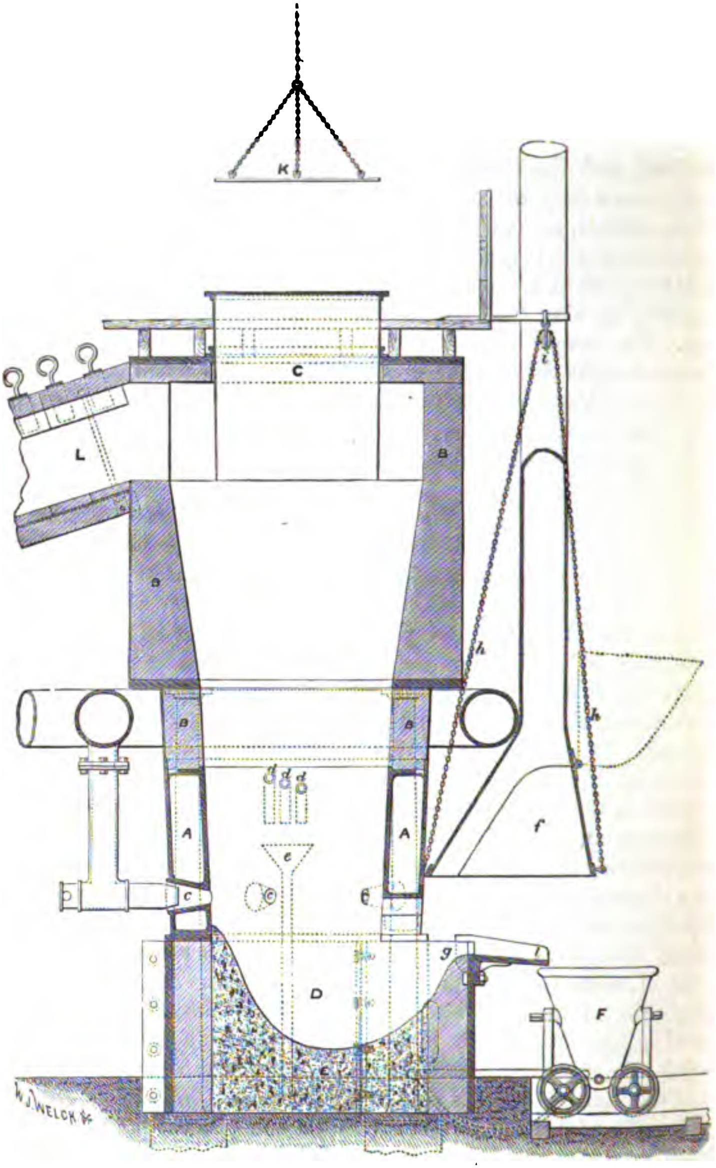

Fig. 187.—Water-jacket Furnace, Couëron; vertical section.

house. At the expiration of about two hours the whole of the charge will have become agglomerated, and is withdrawn. A fresh charge is now let down from the hopper, spread evenly over the sole as before, and the operation of roasting is repeated. In this way three charges of 1,400 kilos. each are worked in twenty-four hours; two men are employed per shift of twelve hours, and the consumption of coal amounts to 22 per cent. of the weight of raw ore treated.

*b. Reduction of Calcined Ores and Grey Slags.*—The form and arrangement of the blast-furnace employed at Couëron will be understood from the woodcuts, figs. 187 and 188; of which the first is a vertical section through the fore-hearth, and the second a horizontal section through the tuyers, but without the slag-lip, *l*. It is considerably higher than the ordinary Castilian furnace, and is charged at the top; the brickwork of the lower portion of the furnace, which is usually burnt through very rapidly, being replaced, to a height of about 3 feet, by an annular cylinder of iron, A, kept cool by the circulation through it of a constant stream of water. The whole of the upper part, B, is cased with strong sheet-iron, riveted together so as to prevent the escape of fume.

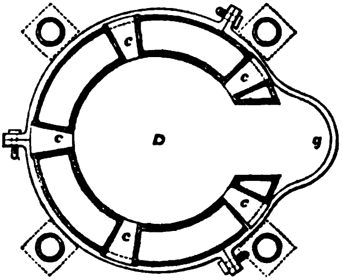

Fig. 188.—Water-jacket Furnace, Couëron; horizontal section.

The body, A, is either made of cast-iron in one piece, with five tuyers, *c*, equidistant from one another, and converging towards the vertical axis of the furnace, or of ½-inch boiler-plate. The number of tuyers, and their position, are matters of importance as affecting the working of the furnace. Were fewer tuyers employed, and were they placed farther from the breast, as is generally the case, the cooling effect of the water in front would be prejudicial to the regular descent of the charge. In this furnace as much as 25 to 30 tons of stuff are smelted in twenty-four hours, the slags being reduced at the same time to a yield of ½ to ¼ per cent. of lead.

In making the hollow jacket, A, it is important not only that the tuyers should be equidistant from one another and converging towards the central axis of the furnace, but also that they should be perfectly round, and have their axes placed horizontally. During the working of this apparatus attention must be paid to keeping up a regular supply of cold water.

The arrangement shown in dotted lines on the vertical section, fig. 187

is the most convenient for the escape of the warm water, since it enables the workmen to see at a glance the quantity and temperature of the water as it flows from the outlet-pipes, *d*, and falls into a funnel, *e*, placed beneath and in connection with a drain. On the side opposite that from which the warm water escapes an inlet-pipe is fixed, which conducts cold water to the bottom of the jacket; this pipe is about 1½ inch in diameter, and is provided with a tap to regulate the supply.

The sheet-iron hood, *f*, shown in fig. 187, is placed over the fore-hearth, *g*, for the purpose of carrying off the fume which escapes from the breast, and might injuriously affect the health of the charger working on the top. This hood is so arranged as not to interfere with the work at the fore-hearth, since, when necessary, it can be pulled up on its hinges out of the way of the workmen by a chain, *h*, and pulley, *i*; the iron cylinder with five tuyers can be adapted to the ordinary furnace, charged at the side or back in the usual way. It is nevertheless thought that this system of charging on the top through the iron cylinder, C, closed by the cover K, is to be preferred for lead furnaces to the old plan, as preventing, to a great extent, the entrance of atmospheric air into the flue. The products of combustion, &c., escape by the flue, L, and the bottom of the furnace, D, is cut out of a mass of brasque, E, in the usual way.

The adoption of the water-jacket has led, at Couëron, to a considerable saving in repairs, bricks, clay, iron, and wages. The most important economy, however, is in smelter's wages. Owing to the fact that the time required at the old furnaces to patch up the sides is now entirely employed in smelting, a proportionately larger quantity of stuff is worked through. The greater regularity in the working of this furnace has the effect of rendering the slags poorer than they were formerly. There is, therefore, not only a saving of lead from this source, but also a reduction of cost consequent on there being no rich slags to re-work.

The quantity of siliceous ores smelted at Couëron being relatively small, it is more convenient and economical to mix them with grey slag, fume, &c., than to treat them alone. The following is the usual composition of the mixture smelted:—

| | Kilos. |
|---|---|
| Grey slags from reverberatory furnaces . | 10,000 |
| Calcined ore . . . . . . . . | 2,000 |
| Agglomerated fume . . . . . | 2,000 |
| Iron slags . . . . . . . . . | 3,000 |
| | 17,000 |

Before lighting this furnace the iron casing is filled with water by opening the cock at the back, and a small fire only is at first made, with wood and coke, for the purpose of drying and heating the bottom, E. This fire is kept up until the bottom becomes perfectly dry and is sufficiently hot to prevent the first slags from setting and reducing the capacity of the breast-pan. Coke is now charged at frequent intervals, until the furnace contains a mass of burning fuel to a height of 3 feet above the tuyers.

The natural draught is at first sufficient to carry on combustion, but when the mass has become thoroughly ignited a light blast is introduced through the tuyers. Before turning on, however, the blast from the fan, the fore-breast is cleaned out, and some burning coke is pulled forward to the level of the lip, *l*; this is covered with a shovelful of cinders, and the opening closed by stiff clay. A hole is made with a pointed bar between the cinders and the clay, so as to allow a jet of flame to escape for the purpose of heating the fore-breast and preventing the first outflow of slag from adhering to the front. After a few minutes this flame is stopped by putting a lump of coke before the hole, together with two or three shovelsful of cinders. Should these not suffice, some clay is pressed down on the cinders to increase their cohesion. The tap-hole is likewise left open for a short time after the blast has been turned on, and is afterwards closed with stiff clay.

The furnace is at first charged with slag only, until fused slag begins to appear before the tuyers. The charging of the furnace mixture, alternately with the requisite proportion of coke, is then begun and subsequently continued with but little interruption, until the furnace has become filled to the height of the hopper, C, below which the charge must not be allowed to sink. The mixture to be smelted is introduced equally all round and against the sides in such a way that a depression is left in the middle after each charge for the reception of coke. From the moment the charge has attained a proper height in the furnace, the blast is increased to its full force, the tuyers being worked with a very short nose of slag, and kept constantly bright.

During the heating-up of the furnace the water in the iron casing becomes hot, and must be constantly renewed by the regular admission of cold water through the feed-tap. This supply is so adjusted that its temperature at the outflow may be always under the boiling-point, or about 80° C.; but comparatively little water is required after the slag has come down, since a crust of the latter adheres to the sides and interposes a layer of non-conducting material between the iron and the fuel in the furnace.

The fused products in their descent collect in the cavity, D, in the bottom of the furnace, formed of brasque, and arrange themselves according to their specific gravities. Lead being the heavier, sinks to the bottom, whilst the supernatant slag flows out through the opening made with a bar in the stopping of the fore-breast, and thence over the lip, *l*, into the cast-iron waggon, F. These waggons are made to run either on cast-iron plates or on a railway to a distance from the furnace, where, on cooling, the slag is tipped out and examined for lead previous to being thrown away.

The flow of slag from the furnace now becomes continuous, or nearly so, being only checked by the gradual cooling of the stream as it runs over the lip. The solidified slag is, however, from time to time detached with a slice, and its flow maintained unchecked until it is found, by sounding, that the lead in the breast-pan has risen nearly to the level of the lip. A bar is then forced into the tap-hole for the purpose of breaking the clay stopping and drawing off the lead into a pot sunk in the floor. At the same time the slag-opening in the breast is temporarily stopped with ashes or clay to prevent the blast from escaping on the lowering of the slag-level, which follows the tapping of the lead.

As soon as slag begins to make its appearance the tap-hole is immediately stopped, either by an iron bar or by a plug of clay stuck on the end of a pole and adroitly wedged into the aperture, while the lead collected in the pot, after having been cleaned and skimmed, is laded into pig-moulds. For a short time after tapping, the wind is shut off, the fore-breast is opened, and the bottom cleaned from any lumps of slag, which, if allowed to remain, would interfere with the working of the furnace. This cleaning-out having been accomplished, in a few minutes the breast is again closed with clay and cinders, and the blast is turned on as before. Shortly afterwards, as the slag rises above the level of the outflow, a hole for its escape is again made in the breast by an iron bar.

It is observed that this furnace generally works faster and better for some time after the withdrawal of the unfused matters accumulated on the hearth bottom; but these are not generally cleaned out after every tapping, unless the furnace is driving slowly; two or three cleanings per shift are usually sufficient, but this depends on the working of the furnace and the nature of the material smelted. The number of lead tappings per day depends on the size of the breast-pan and on the richness of the matters treated. Ordinarily, four or five tappings, each of 16 to 20 pigs, are made per shift of twelve hours.

The average quantity of stuff smelted in twenty-four hours is from 22 to 25 tons, which can be increased to 30 tons with free-running slags. A smelting campaign lasts three months without interruption, and would probably last six, or even twelve months, were the supply of ore unlimited. Attention and experience on the part of the smelter are required to avoid mishaps, and to prevent a premature stoppage of the furnace. In any case, there is no necessity to stop for ordinary repairs, as in the case of Castilian furnaces built entirely of brick.

The consumption of gas-coke of inferior quality is 10 per cent. of the furnace mixture. No iron is added, and no matte is obtained. A little iron is reduced in the furnace from the iron slags employed as flux, and perhaps this accounts for the poorness of the slags produced, which very seldom exceed ½ per cent. for lead. The slags are sometimes so highly basic that it is necessary to add sand to the mixture in order to render it sufficiently fusible, and to prevent the furnace from gobbing. The most fusible slags contain about 30 per cent. of silica. The number of men employed at this furnace is five, and the loss of lead, exclusive of fume collected in the flues, is 6 per cent. It should be mentioned that the furnace is connected with long flues and condensers for the condensation of lead fumes.

Furnaces of this class are, in addition to the ordinary tap-hole, now generally provided with a "syphon-tap," a contrivance by which the molten lead rises through an oblique channel to the bottom of a reservoir in which it rises to the same level as that which it occupies in the furnace. As the lead accumulates in this basin it is laded into moulds, while the slags flow constantly off through an entirely different orifice.

LEAD-SMELTING IN COLORADO.[1] *Ores.*—The material treated at Leadville, Colorado, consists of argentiferous lead ores, which are locally divided into two general classes, namely, *sand carbonates* and *hard carbonates.* The first consists of loose sandy masses of lead carbonate containing chloride of silver, while the second are a porous siliceous material more or less mixed with the hydrated oxides of iron and manganese, associated with lead carbonate and silver chloride; sometimes containing considerable quantities of unaltered argentiferous galena. With the exception of mechanical mixtures of clay, limonite, and silica, they contain but few foreign ingredients. Pyromorphite and anglesite are usually present in small quantities, as are also oxidized compounds of copper, arsenic, and antimony. Ores rich in manganese are usually poor in silver.

The galena is frequently covered by a coating of lead carbonate, indicating the conversion of the sulphide, first into sulphate, and subsequently into carbonate. Bismuth and vanadium occur in the ores from a few of the Leadville mines. In addition to chloride of silver, the oxidized ores contain silver in combination with iodine and bromine. Large masses of hornsilver are occasionally found, and on various occasions several tons of ore have been taken from the R. E. Lee Mine assaying from 8,000 to 15,000 ounces of silver per ton, and almost entirely free from lead. The Leadville ores usually contain so small a proportion of gold that its presence can only be discovered by an examination of the silver obtained, but not in the ore itself. In 1880 the daily output of the several mines was between 700 and 800 tons of ore.

*Fuel and Fluxes.*—The fuel employed is coke and pine charcoal, but before the opening of the railway coke had to be brought over distances varying from 30 to 150 miles by ox or mule teams, and charcoal was consequently more generally used than it is at the present time. The coke employed is made from coal of Upper Cretaceous age, containing from 9½ to 22 per cent. of ash, and is brought from South Park by the Denver, South Park, and Pacific Railroad, or from El Moro by the Denver and Rio Grande Railroad. Coke from Pennsylvania is also used.

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[1] For more detailed information on this subject the reader should consult an article on 'Lead-Smelting at Leadville,' by S. F. Emmons, in 'Precious Metal Deposits of the United States,' Washington, 1885. In this paper, from which the illustrations and data here given have been principally derived, the author acknowledges great obligations to an unpublished report by Mr. A. Guyard.

The fluxes used are limestone, hæmatite, and limonite. The limestone, which is of Cretaceous age, is brought from Cañon City, a distance of 117 miles by rail, while the hæmatite and limonite are obtained from local mines, and contain small quantities of silver. In many cases the ores themselves are so ferruginous as to require a very small addition only of iron oxide. According to Mr. Guyard, the average proportion of flux employed at eight principal smelting-works in the district in the year 1880 was 33·8 parts to 100 parts of ore.

*Furnaces.*—Blast-furnaces are exclusively employed at Leadville, and of these two varieties were in use during the year 1880, the one presenting a circular horizontal cross section, sometimes called the Pilz furnace, while the other is a modification of the Rachette furnace, and is rectangular. In 1882 the rectangular furnace had entirely superseded the former. The circular section, although possessing advantages with regard to the regularity of the descent of the charges, is more expensive in construction, while the diameter of the hearth is limited by the strength of the blast employed.

With ordinary blowing-appliances, a circular furnace can only be successfully worked when of very moderate dimensions. The rectangular furnace, on the contrary, can be constructed with a width at the tuyers commensurate with the strength of the blast, while the production may be increased by extending the length of the cross section. In lead-smelting, it is undesirable to employ the high-pressure blast required by a circular furnace of large diameter, since the resulting high temperature would promote the production of metallic fumes, which are never perfectly recovered.

The rectangular and circular furnaces at Leadville are both constructed on the same general plan with regard to height, method of support, water-jacket, &c., but their dimensions differ very considerably, since their smelting capacity in the twenty-four hours varies from 15 to 40 tons. Fig. 189 represents, in elevation, the rectangular furnace employed at Leadville, drawn to a scale of 9 feet to the inch. Fig. 191 is the same furnace in horizontal section at the level of the tuyers; fig. 190 shows a vertical section on its larger diameter; and fig. 192 a horizontal section at the height of the charging-door. The stack of refractory masonry, *a*, rests upon a cast-iron plate, *b*, supported upon pillars, *c*, in order that no unnecessary weight may rest upon the bottom portions of the structure. The sides and ends of the hearth for some distance above and below the tuyers are water-jackets, *d*, formed in sections, and constructed either of cast-iron, wrought-iron, or steel. In the furnace of which an illustration is given the water-jackets are twelve in number, firmly bolted together and provided with openings for the introduction of the tuyers, *e*. A cold-water pipe runs around the furnace above the water-jackets, and water is admitted into each by a suitable tap; outlets for the heated water, and gutters for its removal are also provided. The interval between the water-jackets and the cast-iron plate, *b*, which supports the shaft of the furnace, is filled with brickwork, which, in case of necessity, can be easily removed. The lower ends of the water-jackets rest upon the hearth, *f*, which consists of cast-iron plates bolted together and lined either with fire-bricks or brasque, which is locally known as "steep." The usual form of the

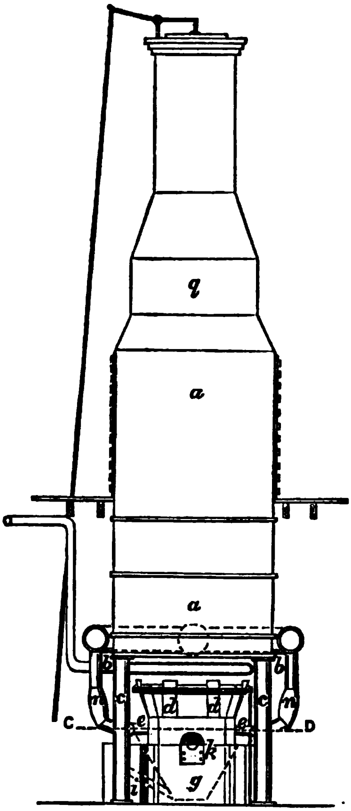

Fig. 189.—Elevation.

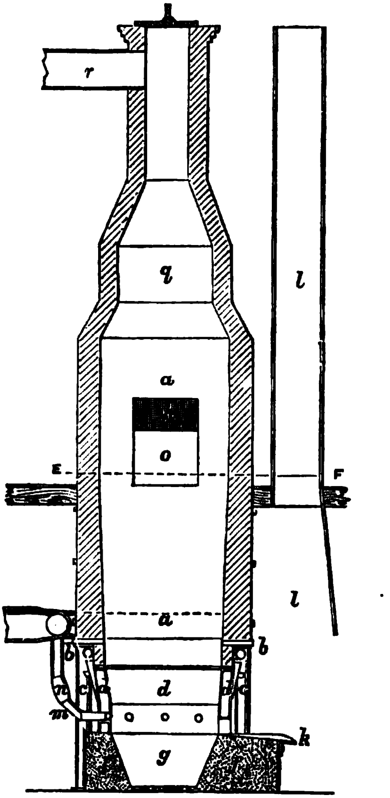

Fig. 190.—Vertical Section.

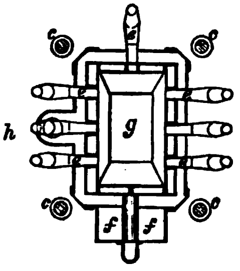

Fig. 191.—Section on C, D.

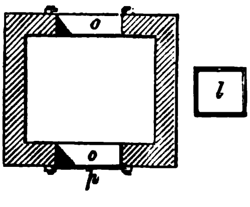

Fig. 192.—Section on E, F.

Rectangular Furnace, Leadville.

bottom, *g*, is shown in the illustrations, but this may be modified according to the working of the furnace without involving any change in the casing of iron plates. The hearth-plates enclose the lead-well, *h*, and the channel, *i*. The melted lead, rising through the oblique passage, *i*, fig. 189, to the same level which it occupies in the furnace, can be laded into moulds at leisure from the lead-well, *h* (fig. 191), without the interruption to the work incident to the old method of tapping at intervals.

At the end of the furnace, just above the hearth, an opening is left in the water-jacket for tapping-off the slag. This is stopped with clay, which can be pierced when required, allowing the slag to run off through the gutter, *k*, into a slag-bogie placed beneath for its reception. A hood, *l*, is generally placed over the tapping-hole to carry off any metallic fumes which may be given off during the operation. The number of tuyers varies with the dimensions of the furnace, but a tuyer is always placed at the extremity of the hearth opposite the slag-tap. Sometimes a tuyer is also introduced above it, but this is somewhat in the way, and is consequently often omitted. A slide, *m*, at the elbow of the nozzle, admits of looking into the interior of the furnace, and the introduction, when necessary, of a small iron bar. The tuyers are connected to the blast-pipes, when cold-blast is employed, by canvas hose, *n*, the flexibility of which admits of their adjustment or removal when required. The feed-openings, *o*, on the upper floor, are closed by counterpoised doors, *p*. The furnace terminates in a short chimney, *q*, and may, in case of need, be worked without the dust-chambers, with which it communicates by the flue, *r*.

As an example of furnaces of this description recently erected those of Eddy, James, and Grant, built at Denver in 1882, may be cited.[1] There are eight of these furnaces of the same pattern and size, of which the height to the charging-floor is 18 feet. Their diameters 3 feet below this point are 60 × 102 inches; at the top of the jackets 48 × 92½ inches, and at the tuyer level 36 × 80½ inches. The smelting-capacity of each is about 30 tons per twenty-four hours. Each furnace has ten tuyers, four on each side, and one at each end. The fumes and waste gases are drawn from them, a little below the charging-floor, into large dust-chambers connected with a high stack. The blowing-apparatus employed at Leadville is usually Baker's rotary blowing-engine, although at one establishment a Roots' blower is used.

*Smelting.*—The furnace is first dried by a slow fire of wood or charcoal, and, when dry, the fire is allowed to burn out and the brickwork left to cool. The hearth is then lined with brasque in the usual way, the dam and syphon-tap being frequently lined with fire-clay only. The shaft is filled to the height of the feed-door with charcoal, the tuyer-holes and tap-hole being left open to create a draught, and the whole gradually becomes incandescent to the throat. The blast is then gradually turned on; the tuyer-holes in the water-jackets, with the exception of either two or four, are stopped with clay, and the canvas hose of the corresponding tuyers tied with string. Tuyers are inserted in the holes left open, the tymp is set in its place, and the blast turned on at full pressure. A long jet of flame now issues from the syphon-tap, and the blast is steadily kept up until the lead-well has become thoroughly red hot. The remaining tuyer-holes are afterwards opened and the blast admitted at the normal pressure; the hearth is now ready for filling.

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[1] 'Engineering and Mining Journal,' xxv., p. 163.

Pigs of argentiferous lead, kept in reserve for this purpose, varying in amount from 4 to 12 tons, in accordance with the capacity of the furnace, are now introduced, together with additional fuel, the proportion being about 14 lbs. of charcoal to 100 lbs. of lead. From 100 to 150 bushels of charcoal, according to the capacity of the furnace, are consumed during the blowing-in; when the lead makes its appearance in the well it is covered with a few pieces of charcoal, and the furnace is ready for work.

Old slags are first thrown into the furnace, which is not in a condition for charging until these have become perfectly liquid. The tap-hole in the tymp is opened from time to time, and the regular charging begins only when the slags are found to flow freely. In charging, the ores, slags, and fluxes are mixed together, and are deposited next the walls, while the fuel is thrown into the depression left in the centre. The tapping of slags begins as soon as the furnace gets into regular operation, and usually takes place at intervals of fifteen or twenty minutes, although in a few works the system of a continuous flow is adopted. From time to time the metal is dipped from the lead-well in wrought-iron ladles, and is poured into moulds. The pig lead, or *base bullion,* carries from 100 to 250 ounces of silver per ton of 2,000 lbs., and from 0·25 to 4 ounces of gold.

The length of a run is seldom less than three weeks, and sometimes extends over as long as thirteen months. According to Mr. Guyard, the average consumption of fuel is 32·83 per cent. of the ore worked, or 23·6 per cent. of the mixture put through the furnace. Of the lead present in the ores 88 per cent. is obtained in the metallic state, the remainder going into the slags or escaping into the stack. Each unit of lead produced necessitates the expenditure of 1½ unit of fuel, and the cost of smelting varies from $12 to $18 per ton (2,000 lbs.) of ore treated.

The "base bullion" produced in Colorado is forwarded to the Eastern States, where it is treated for the silver and gold which it contains.

HORNO DE GRAN TIRO, OR PAVO.—This furnace, which is used in some parts of Spain, and particularly in the district of Cartagena, for smelting poor carbonates of lead, is a barrel-shaped slag-hearth, which, instead of being supplied with blast by a fan or blower, is worked by a strong draught obtained by the aid of a chimney of about 45 feet in height. This is connected with the furnace by an inclined flue at the top, while the air enters through six horizontal tuyers, of refractory clay, arranged radially around it a little above the level of the hearth, which is formed of brasque in the usual way. Coke is the fuel employed, and the ores treated commonly yield, by assay, about 12 per cent. of lead; the average production of metal is from 8 to 9 per cent., and the consumption of fuel from 30 to 32 cwts. per ton of lead obtained.

The methods of treating lead ores in the blast-furnace, and the form and dimensions of the furnaces employed, vary in accordance with the nature of the ores and the description of the fuel available, and admit of almost endless modification. In this country, where the ores treated are delivered to the smelter in a concentrated state, and where fuel is comparatively cheap, the blast-furnace has, until lately, been seldom employed, except for the reduction of slags and other furnace products.

## Smelting in Shallow Hearths.

Backwoods Hearth.—The early settlers in Missouri, United States of America, were in the habit of extracting lead from galena by means of a rude hearth constructed of earth and rough stonework, but without the aid of any sort of blast. The front wall of this hearth was about 8 feet in length and 6 feet in height; the internal cavity was 8 feet long and 2 feet wide at bottom, but gradually widened towards the top. This sloped regularly towards the front wall, in which, on the prolongation of the longer axis, was an arch enclosing an aperture through which the molten metal made its escape.

In the bottom of the cavity thus formed, a layer of heavy logs was laid, longitudinally, and then followed a stratum of split billets; upon these the galena was deposited, and the whole covered by a layer of smaller branches chopped into short lengths. The fire was kindled through the front arch, which, with the exception of a hole for the escape of metal, was subsequently closed, and the reduced lead, flowing continuously through it, was collected in a basin in front. The time occupied by this operation was twenty-four hours, and nearly pure galenas afforded one-half their weight of metal; the lead and slags remaining in the ashes were subsequently treated in a rough substitute for the slag-hearth called an "ash-furnace." This very primitive method of smelting, although of comparatively recent date, is now obsolete, but is probably very similar to some of the processes employed in remote antiquity for the extraction of lead from its ores.

Ore-Hearth, or Scotch Furnace.—In the North of England the smelting of lead ores is often conducted in the ore-hearth or Scotch furnace. This consists of a rectangular cavity about 22 inches square; its depth varies from 22 to 26 inches, the whole of its internal surface being lined with cast-iron. The bottom, which consists of but one casting, is surrounded by a ledge; excepting, sometimes, on the side facing the *work-stone, a*, fig. 193, which may be 2 feet 10 inches in breadth, and about 1 foot 10 inches in the other direction.

This is provided with a shallow ledge, *b*, on each side, and is placed with a fall of a few inches on its length. Its higher side, *c*, rests on the hearth bottom, or in some instances is united to it, and forms only one casting. When this is not the case, the joint between the two is closed and made lead-tight by a cement of moistened bone-ash well kneaded together. On the back edge of the bottom is placed a prism of cast-iron called a *back-stone*, about 6½ × 5 inches, and 28 inches in length; on this rests the nozzle of the tuyer, over which is again placed another iron casting called the *pipe-stone*, of the same length as the back-stone, and 10 inches square. This has, at the centre, a cavity for the introduction of the tuyer. On it is again placed another back-stone of nearly the same dimensions as the first, which completes this side of the hearth, and usually makes its total height from the bottom about 25½ inches. Along the lateral edges of the bottom are placed two prismatic castings called *bearers;* these are each 26 inches in length and 7 inches square, which project slightly over the upper edge of the work-stone. Above these bearers, and at a distance of 13 inches from the back of the hearth, is supported another bar of cast-iron called the *fore-stone*, which has the same form and dimensions as that on which rests the tuyer of the blowing apparatus. The space at each end of the fore-stone is now closed by two lumps of cast-iron measuring 10 inches of a side, called *key-stones.*

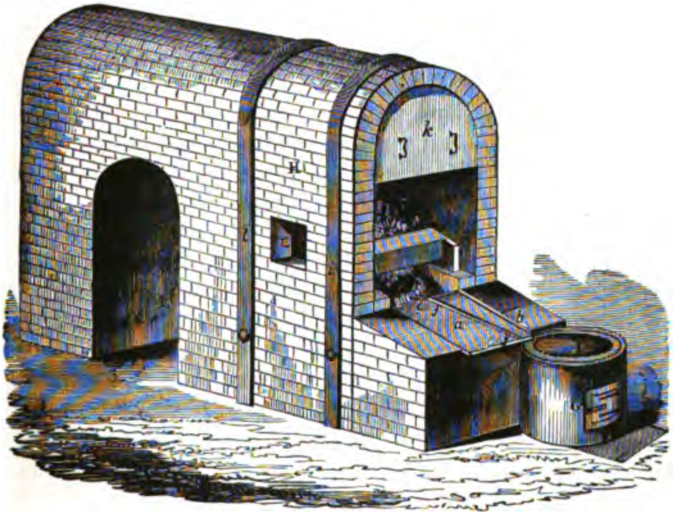

Fig. 193.—Ore-Hearth.

Before the work-stone, *a*, and set in masonry enclosed in a circular cast-iron casing, G, is the lead-pot, E, into which the melted metal, as it issues from the hearth, is conducted by the oblique channel, *f*, sunk beneath the surface of the iron plate. In the woodcut this pot is not shown sufficiently near the furnace. To prevent the escape of fumes into the smelting-house, which might injure the health of the persons employed, the entire hearth is often enclosed in a hood of arched masonry, H, communicating with the chimney, and in which is left a small door, I. In the majority of ore hearths the hood is carried much higher than that shown in the woodcut. The sheet-iron door, *k*, admits of being raised or depressed at pleasure, according to the degree of draught required; and the blast communicating with the tuyer is regulated by a valve placed in a pipe approached by the arched opening, L, left for that purpose. The brickwork is consolidated and bound together by the iron straps, *l*, kept in their places by bolts passing beneath the foundations of the hearth.

*Roasting.*—Formerly the ores smelted in the Scotch furnace were subjected to no other preparation than careful dressing previous to their metallurgical treatment. It is, however, sometimes found advantageous to roast them, so as to effect their partial desulphurization and oxidation, before treating them for the metal they contain. The furnace employed for this purpose varies considerably in its dimensions in order to suit local circumstances, but always consists of a flat hearth, covered by a low arch, and is heated by a fire-place situated at one end; there are also doors on either side, for the withdrawal and working of the ore. From 10 to 14 cwts. of galena constitute the charge of a furnace of this description, and require from two and a half to three hours to become sufficiently roasted. The mineral, after being introduced into the furnace, is first spread evenly over the bottom, and the fire so arranged as to keep it constantly at a temperature below the melting-point of galena. Copious sulphurous fumes escape from its surface, and if any portion should, from approaching too nearly the point of fusion, become softened, fresh surfaces are presented to the air. In this way a portion of the sulphur is driven off, and slimes and other friable substances are so agglutinated as to resist the blast without being carried off as dust into the flues.

*Smelting.*—At the termination of each shift the hearth is supposed to be in working order and the bottom to be nearly full of lead; a quantity of ore remains on the hearth in a semi-reduced state, called *browse* or *brouse*, and is more or less mixed with fragments of slag and clinker, from which it is roughly separated.

To commence a new shift, the cavity of the furnace is filled with peat cut into the usual rectangular blocks. Those at the back part of the hearth are heaped up without any kind of order, but those placed towards the front are arranged in the shape of a regular wall. The bellows or fan is now set in action, and an ignited peat is thrown before the nozzle, which quickly communicates the fire to the whole mass. On the top of this a few shovelsful of coal are afterwards sprinkled, for the purpose of binding and consolidating the mass, as well as to increase the temperature. The browse resulting from the preceding operation is then thrown on the surface of the ignited mass; and shortly afterwards the material contained in the hearth is stirred with a poker, and a portion of it drawn out on the work-stone. The slag is now removed with a shovel, and thrown aside for subsequent treatment. The browse, cleaned from slag, is again thrown back into the hearth, with the addition, if it be required, of a little coal. If, as sometimes happens, the browse has not been properly freed from slag, but becomes pasty and evinces a tendency to fuse, it must be hardened by the addition of lime, which dries the mixture and facilitates the subsequent extraction of lead. When, on the contrary, the ore is found too refractory, a small addition of lime is also made; but in this case a less quantity is employed, as it is only intended as a flux for the refractory matters present, and not, as in the other instance, to act as a dryer of the too fusible scoriæ obtained. The lumps of slag thus formed contain a considerable portion of the lead present in the ores, and are therefore collected for the purpose of being treated in the slag-hearth.

When the whole of the browse has been thrown back into the hearth, a few shovelsful of ore are thrown on the top of it; before doing this, however, it is necessary to place a lump of peat before the tuyer, which not only prevents any of the mineral from entering the nozzle, but likewise serves to spread the blast equally through the mass. After an interval of about twenty minutes, the contents of the furnace are again partially drawn out on the work-stone, and another portion of metallic lead is carried by the channel, *f*, into the pot, E. The grey slag is removed, and another lump of peat is placed before the tuyer. The browse, together with a proper quantity of coal and lime, is again thrown on the hearth fire, and on the top of the whole a fresh supply of raw or roasted ore. Two men are employed on each shift, the operations being continued during from twelve to fourteen hours; at the expiration of that time, a production, varying with the nature of the ore, of from 22 to 30 cwts. of metallic lead is obtained.

Towards the end of the shift no addition of ore is made, and, after stopping the blast, the browse is taken out, and roughly separated from slag. The bottom of the hearth is now filled up with lead from the receiving-pot ready for the next shift, and in this way the charge is constantly kept floating on a bath of metallic lead. At the end of a shift of from twelve to fifteen hours, the hearth, in spite of every precaution, is liable to become too hot, and smelting is usually suspended five hours after each shift.

The lead prepared by this process is said to be purer than that produced in the ordinary smelting-furnace. This may arise from the circumstance that, being exposed to a less elevated temperature, the more fusible constituents of the ore are alone obtained, while in the smelting-furnace the heat employed is so great as to effect the reduction of some of the foreign metals contained in it, which, by entering into combination with the lead, tend to impair its quality. When ores, assaying from 75 to 80 per cent., are operated on, from 1½ to 2 cwts. of coal and about four small cartloads of peat are required to produce a fodder (21 cwts.) of lead. The cost of lead-smelting in the ore-hearth is generally in excess of that by the reverberatory furnace, but with rich ores the loss of lead is less considerable. The ore-hearth is now generally worked with coal as the only fuel, the use of peat being abandoned.

THE AMERICAN HEARTH.—This hearth, first introduced at Rossie, New York, and subsequently used in Missouri, has a cast-iron bottom, 24 inches square, 12 inches deep, and 2 inches in thickness. The work-stone, which is 32 inches wide and 22 inches from front to back, is provided with raised sides, and has the usual diagonal groove for directing the reduced metal into the receiving-pot. An air-chest, 14 inches in height, of cast-iron, forms a wall on the sides and back of this hearth; its outside width is 6 inches, and, as the thickness of the metal is ¾ of an inch, a vacant space is left within, a little more than 12 inches in height and 4½ inches in width. The blast passes into this box on one side, and escapes at the other through a curved pipe which conducts it to a tuyer occupying the usual position at the back of the furnace. In this way the sides and back are kept cool, while the blast is heated before entering the fire. The bottom is, like that of the ordinary ore-hearth, kept full of molten lead, on which the charge floats during the operations of smelting. The fuel employed is wood or pine charcoal, and the galena treated must be broken into pieces, which should not be larger than ⅛ of a cubic inch. This hearth remains continuously in blast during six days of the week, and is worked by four men, two on each shift. About a quarter of a cord of wood is consumed per ton of lead obtained, or the wood burnt reduces a little more than 2½ times its weight of metal. The daily consumption of wood in a hearth of this description is three-quarters of a cord, and the yield of lead is about 7,500 lbs.

An experimental ore-hearth on this principle was in operation at Bleiberg, in Carinthia, during the years 1849, 1850, and 1851, and it was also tried at Przibram in 1856; but has not been permanently adopted at either place.

## SOFTENING AND DESILVERIZING PROCESSES.

The *work-lead* obtained from the smelting-furnace in addition to silver, usually contains antimony, copper, and other oxidizable impurities, and these are sometimes present in sufficient quantity to materially interfere with the desilverizing processes proper; they are, therefore, when necessary, previously removed by a process known as *improving* or *softening*.

IMPROVING OR SOFTENING.—This operation consists of fusing the lead in a reverberatory furnace of peculiar construction, and allowing it to remain, when in a melted state, exposed for a more or less considerable period to the oxidizing influences of the air. By this treatment the antimony and other metals, together with a portion of the lead, become oxidized, and are removed from the surface of the bath by an iron rake; thus constantly exposing a fresh surface to the action of the heated gases, until the greater portion of the impurity is removed, and a nearly pure alloy of lead and silver is obtained.

The hearth of the furnace in which this operation is conducted often consists of a large cast-iron pan about 2 inches in thickness, which may be 10 feet in length, 5 feet 6 inches in width, and 10 inches in depth. Wrought-iron pans are also used. The fire-place, which is about 20 inches in width, has a length equal to the width of the pan, from which it is separated by a low bridge 2 feet in width. The arch at the bridge end is 16 inches above the edge of the pan, while at the other extremity its height from the same point is 8 inches only. All the angles of the casting are carefully rounded in order to prevent breakage from expansion or contraction, and the softened lead is, when required, drawn off into a cast-iron pot by a hole bored in the bottom near its outer edge. This, when necessary, is stopped by a well-fitting iron plug kept in its place by a weighted lever.

The charge, which is about 11 tons, is first fused in an iron pot set in brickwork at the side of the furnace, and is subsequently laded into the pan through a sheet-iron gutter prepared for the purpose. When the metal is in a fit state for tapping, a small portion taken out in a ladle, and poured into an iron mould, will be observed on cooling to present on the surface a peculiar flaky subcrystalline appearance, which, when once seen, is again easily recognised. As soon as this appearance presents itself the fire is lowered, the plug loosened, and the contents of the pan are drawn off into a pot, from which they are afterwards laded into moulds.

In some cases, as at Pontgibaud, much larger furnaces than that above described are employed for the operation of softening. At that establishment the *improving pans* are each 13 feet in length by 6 feet 6 inches in width, and are capable of working charges of 20 tons.

The time necessary for softening hard lead necessarily depends on the proportion of impurity it contains. Consequently some varieties will be sufficiently purified after the expiration of twelve hours, while in other cases it becomes necessary to continue the operation during several days. Ordinary hard lead from the Cornish flowing furnace, or from the Castilian blast-furnace, is usually softened in about thirty-six hours, with a consumption of about 2 cwts. of coal per ton. In some smelting establishments the hard lead is softened in an ordinary reverberatory furnace, provided with a slag bottom. When a furnace of this description is employed, calcination takes place at a higher temperature, and the operation is conducted more rapidly than in iron pans. The consumption of fuel is about the same, but the loss of lead by volatilization is somewhat greater. In Germany a marl (clay and lime) bottom like that of the refinery furnace is preferred.

The *calcined dross*, removed from the surface of the pan, is treated in the reducing furnace, and the resulting *cinder* in the slag-hearth, or in some other form of blast-furnace. The very hard lead thus obtained is again subjected to a process of softening in the improving furnace, and a calcined dross is ultimately obtained affording a brittle alloy, which is usually sold to type-founders. The quantity of this very hard lead annually produced, even in large establishments, amounts to only a few tons. At Freiberg 270 tons of hard lead of all kinds are made from 10,000 tons of furnace-lead treated annually.

LIQUATION.—At Freiberg and Przibram, coppery lead is subjected to liquation before softening. A reverberatory furnace is used, having a marl bottom, which slopes steeply from the fire-bridge and terminates in a sump or lead-pot at the flue end. The pigs of furnace-lead are piled at the upper end of the bed and subjected to a very low heat, when lead separates and flows down into the sump, whence it is laded into moulds, after skimming, leaving about 6 per cent. of refractory dross (*Saiger dörner*) behind, which contains nearly the whole of the sulphur, copper, and nickel, as well as about one-fourth of the arsenic present in the furnace-lead. About 13 tons of lead are passed through the liquation-furnace in twenty-four hours.

## SEPARATION OF SILVER FROM LEAD.

Before the discovery, by Hugh Lee Pattinson, of the process by which the silver in argentiferous lead may be concentrated in a comparatively small amount of that metal, the whole of the lead obtained by smelting was, when sufficiently rich, subjected to cupellation. When, however, it contained less than about 8 ounces of silver per ton it was not considered to pay the expenses of the operation; whereas, by the method of crystallization, lead containing only 2 ounces of silver per ton may sometimes be desilverized at a profit. A considerable proportion of the lead produced, both in this country and in others, contains a less amount of silver than is necessary to pay the cost of its extraction by direct cupellation, and consequently, until the discovery of the Pattinson process, the whole of this silver was lost to commerce.

This process is founded on the circumstance, first noticed by Mr. Pattinson in the year 1829, that when lead containing silver is melted in a suitable vessel, and afterwards suffered to cool slowly, with constant stirring, at a temperature near the melting-point of lead, small crystals begin to form within the liquid alloy, which, as rapidly as they are produced, sink to the bottom, and on being removed are found to contain less silver than the lead originally operated on. The fluid alloy from which the crystals have been separated is at the same time rendered proportionately richer in silver.

THE PATTINSON PROCESS.—This operation is conducted in a series of from nine to twelve cast-iron or steel pots, which, if worked by hand, may each contain 6 tons of metal, but when cranes are employed 10-ton pots are more generally used; these are ordinarily 5 feet 4 inches in diameter and 2 feet 6 inches in depth. A pot at one end of the series has generally a capacity equal to two-thirds only of that of each of the others, and is known as the *market-pot.*

Each pot is provided with a separate fire-place, and is heated by a circular flue passing round it, which can be closed when required by a damper. The products of combustion finally escape into a flue below the level of the floor, running parallel with the line of pots. In order the more easily to follow the process, we will suppose the lead operated on, which, according to its quality, may or may not have previously undergone the process of improving, to contain about 21 ounces of silver per ton. If the market-pot be called No. 1, this lead will be introduced into No. 6, fig. 194.[1] When fused it is carefully skimmed with a perforated ladle, in order to remove the covering of oxide or *pot-dross*, and the fire is at once withdrawn. The cooling of the lead is now promoted by sprinkling water on its surface, and while its temperature is being thus lowered it is kept constantly stirred with a chisel-pointed iron bar called a *slice.* All those portions which become solidified and adhere to the sides of the pot are removed, and forced under the surface of the liquid metal, where they again become melted. By this treatment crystals soon begin to make their appearance; and in proportion as these form and accumulate at the bottom they are removed by a large perforated iron ladle, in which, after having been well shaken, they are first allowed to drain into the pot whence they have been removed, and are afterwards carried over and deposited in the next pot (No. 5), in the direction of the market-pot. This is continued until two-thirds of the lead in pot No. 6 has been transferred in the form of crystals to pot No. 5; when the lead remaining in No. 6 will contain about 40 ounces of silver per ton, while that transferred to No. 5 yields only about 11 ounces per ton.

The enriched lead in the bottom of No. 6 is now laded into No. 7, next on the left, which eventually becomes filled with lead containing 40 ounces of silver per ton. A fresh supply of lead of the same tenure in silver is now introduced into pot No. 6, and the resulting crystals passed in the direction of the market-pot, while the enriched lead, remaining in the bottom, is laded into the pot next to it on the other side. Each pot in succession, as it becomes filled by crystals from the one side, or by bottoms from the other, is in its turn crystallized.

In this way the crystals obtained from the pots as they go "down the house" towards the market-pot become gradually poorer, while the pot-bottoms passing "up the house" in a contrary direction are progressively increasing in richness. The final result, consequently, will be that at one end of the line of pots the lead will contain but little silver, while at the other it will have become much enriched.

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[1] The system adopted in numbering the pots in different establishments is not always the same. In many cases the numbers are made to commence with the pot next the market-pot, which is called No. 1; for the purpose of describing the process it has, however, been considered more simple to begin with the market-pot, which thus itself becomes No. 1.

Any lead that may be on hand assaying about 40 ounces of silver will be introduced into pot No. 7, while lead containing 11 ounces will be melted in No. 5. The other pots in the series may, in the same way, from time to time receive lead yielding the same, or a nearly similar amount, of silver per ton as the metal they severally contain. When this system, which is known as the method of *thirds*, is strictly adhered to, the lead in each pot will be, approximately, twice as rich in silver as that which is next to it in the direction of the market-pot. If, however, a different lead has been introduced into any of the pots, the ratio of this progressive increase in silver may be more or less interfered with. In the richer pots the separation of silver is less complete than in the poorer ones, and consequently the progressive enrichment will not be so rapid. When the lead contains about 2¼ per cent. of silver, or 700 ounces per ton, no further concentration is possible, as both the crystals and liquid lead are then of the same composition. In working the last pot the whole of the bottom is not always laded out, as it is sometimes found advantageous to subject it to a treatment by which the richness of the alloy is still further increased. When the ordinary quantity of two-thirds of the lead has been transferred in the form of crystals to the pot next to it down the house, the remaining one-third will consist of a mixture of crystallized and uncrystallized alloy. The latter being much richer than the former, is separated as completely as possible, and this portion, only amounting to a little more than one-half the bottom, is sent to the refining-furnace. This separation is effected by pressing the mixture with the curved side of one of the large perforated ladles, when the still liquid alloy enters the bowl and is removed by a small unperforated dipper. The lead thus obtained will evidently be richer in silver than the portion remaining in the pot in the form of crystals.

The desilverized metal, or *market-lead*, should not contain above 10 dwts. of silver per ton, while the *rich lead* is usually concentrated so as to contain from 400 to 600 ounces per ton. During the whole of these operations oxidation of lead is continuously going on, and it may be estimated that lead assaying 20 ounces of silver per ton will produce 25 per cent. of its weight of *dross*. At Pontgibaud, where the lead operated on often contained nearly 100 ounces of silver per ton, about one-third of the weight of the market-lead produced was skimmed off the various pots and passed to the reducing furnace.[1]

In this establishment, where a series of twelve pots was employed, the assays of the successive numbers, in 1867, averaged as follows:—

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[1] Before skimming a pot a little sawdust or spent tan is scattered over the surface of the metal, and well incorporated with the scum floating on the top. This facilitates the separation of metallic lead, when the dross is removed in a small perforated ladle, and also, to some extent, acts as a reducing agent during the subsequent treatment of the oxides in the reverberatory furnace.

| | Silver per Ton. | | | |
|---|---|---|---|---|
| | Oz. | dwt. | gr. | |
| No. 1, market-pot . | 0 | 9 | 15 | Market-lead. |
| " 2 . . . . | 0 | 19 | 7 | |
| " 3 . . . . | 1 | 18 | 14 | |
| " 4 . . . . | 3 | 17 | 4 | |
| " 5 . . . . | 7 | 7 | 21 | |
| " 6 . . . . | 12 | 17 | 5 | |
| " 7 . . . . | 22 | 10 | 3 | |
| " 8 . . . . | 36 | 19 | 12 | |
| " 9 . . . . | 59 | 9 | 16 | |
| " 10, charging-pot. | 96 | 9 | 4 | |
| " 11 . . . . | 167 | 3 | 2 | |
| " 12 . . . . | 273 | 6 | 1 | Bottom of No. 12 gave 514 ounces 9 dwts. 18 grs. per ton. |

The ladle employed, when manual labour is made use of, is 16 inches in diameter, 5 inches in depth, pierced with ½-inch holes. When cranes are used, the ladles are 20 inches in diameter, 6½ inches in depth, and are pierced with ¾-inch holes; thickness of iron, ½-inch; length of handle, 9 feet 6 inches. The large baling-ladles used for turning back the bottoms are 14 inches in diameter and 8 inches deep, having a handle 7 feet long.

When, during the operation of fishing out the crystals, the ladle becomes chilled and the holes partially closed, it is heated to the proper temperature by being placed for a few seconds in the pot of hot lead, into which they are turned over. It was formerly usual to provide small pots for this purpose filled with lead, called *heaters*, in front of the larger pots, but these are now always dispensed with. Two crystallizers are employed in working each pot, and one fireman is required for each set; the working of each pot occupies about two hours, and by the use of cranes 10-ton pots can be worked as expeditiously as 6-ton pots by hand.

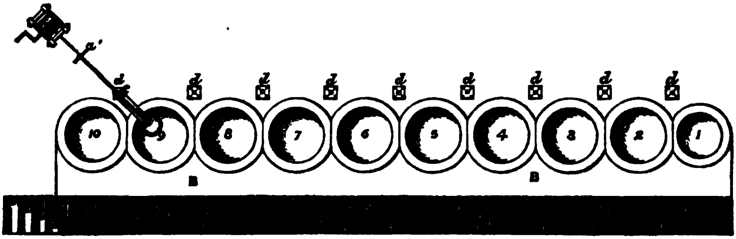

Fig. 194.—Pattinson's Pots; plan.

When the lead operated on contains about 24 ounces of silver per ton, the average expenditure of coal per ton of lead treated is 7·14 cwts. At Pontgibaud the desilverization of the rich work-lead produced was attended with a consumption of about 9 cwts. of coal per ton.

Figs. 194 and 195 represent a plan and elevation of a set of Pattinson's pots arranged for working with cranes. No. 1 is the market-pot, having two-thirds the capacity of the others, which are working-pots. A long ash-pit, A, extends the whole length of the set, and is partially covered by the iron platform, B, supported on pillars; each of the fireplaces, *a*, is provided with an iron door.

In order to desilverize by this arrangement, the potman sinks the ladle sideways to the bottom of the pot, and having turned it over so as to become full of crystals, he attaches a hook to the cross-handle, *a′*, of the ladle, fig. 194, which is then withdrawn by the other workman, who turns the winch. In doing this the iron shank slides over a roller on the front of the crane, *d*, and as soon as it is withdrawn from the metal, the first workman, who guides the handle, slips it into one of the cheeks, *c*, at the back, where it becomes securely fixed. The ladle, filled with crystals, is thus suspended over the pot whence it was withdrawn, and after being allowed to drain for a short time it receives a few shakes by smartly jerking the handle, which for this purpose is released from the cheek at the back of the crane. This is now swung round, the shank of the ladle slipped from under the catch, and the crystals deposited in the pot next on the right. This is continued until the necessary amount of crystals has been withdrawn, when the enriched lead, remaining in the bottom, is taken out by a ladle without perforations, and is turned over in the next pot on the left.

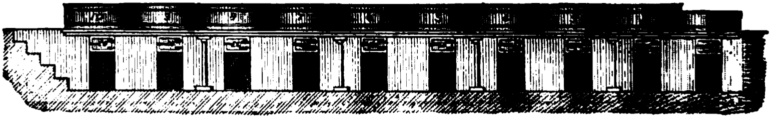

Fig. 195.—Pattinson's Pots; elevation.

Although the method by thirds is that usually adopted for the desilverization of lead moderately rich in silver, no general rule can be laid down with regard to the system of working to be employed, as this may be more or less varied in accordance with the particular requirements of the case. For the desilverization of poor argentiferous lead a system called the method by *eighths* is sometimes employed, when, instead of removing two-thirds of the contents of each pot in the form of crystals, seven-eighths are taken out in that state. The treatment of poor argentiferous lead may, however, be effected by a combination of the two systems; beginning by the method of eighths, the enriched lead may be further concentrated by the method of thirds. The desilverization of lead by crystallization has, however, to a large extent been replaced by desilverization by zinc.

Modifications of Pattinson's Process.—A patent was granted to Mr. P. J. Worsley, in 1860, for "Improvements in the Separation of Silver and Lead," but although sundry experiments carried out at Rotherhithe sufficiently demonstrated the efficiency of the process, it was ultimately abandoned, on the ground of the expensive nature of the necessary alterations in plant, and on account of the time required for the workmen to acquire the requisite experience. A very similar arrangement for effecting the same object was, however, introduced at lead-works at Rouen and elsewhere, under the name of the "*système Laveissière.*" The arrangement employed for this process essentially consists of two cast-iron vessels, the first of which is called the *melting-pot* and the other the *crystallizing-pot*, which must be placed at such a level that the metal from the melting-pot may be run directly into it. Below the level of the crystallizing-pot must be one or more receivers for the reception of the enriched lead. The melting-pot is provided, on the side next the crystallizer, with a discharge-pipe closed by a slide-valve.

The crystallizing apparatus is a cast-iron pot, provided with a vertical stirrer, which, at opposite sides of the bottom, has discharge-pipes fitted with slide-valves. Each of these is heated by a fire to prevent its becoming obstructed by the cooling of the metal within it, and below are placed pots for the reception of the liquid alloy, which, on opening the valves, drains from the crystals retained in the pot above. The stirring apparatus consists of two vertical shafts of wrought-iron working vertically in the centre of the pot; one of these is solid, and stands on a step cast on the bottom, while the other, which encloses it, is a tube, supported by a collar. Each axle has, at its upper extremity, a mitre-wheel, and the outer one being shorter than the other, another mitre-wheel is made to work horizontally between them in such a way as to cause the two shafts to revolve in contrary directions. To the lower extremity of each is attached an iron stirrer provided with knives, which almost touch the sides of the pot, so as to protect them from any incrustation of chilled lead. The object of this stirrer, which receives its motion by a belt, from a steam-engine or water-wheel, is to promote throughout the mass the production of that uniformity of temperature necessary for the crystallization of the metal, and to so compress the crystals formed as to cause them to separate readily from the liquid alloy.

With the formation of increasing quantities of crystals the resistance to stirring becomes greater, and when a certain quantity has accumulated, a considerable amount of power becomes necessary. The apparatus employed should therefore be provided with a tightening pulley, or some similar contrivance, by which the belt is made to slip when the desired accumulation of crystals has taken place. These conditions may be so adjusted as to suit any system of working; but when the liquid alloy is to be reduced to one-third the weight of the total contents of the pot, the stirrer must be arrested as soon as two-thirds of the charge have assumed the crystalline form. The pots are heated by separate fire-places, and above those into which the liquid alloy is run out, is a crane by which the enriched lead is lifted, by means of an iron eye-bolt placed in the mould before the lead solidifies, and is brought back to the melting-pot, to be again treated.

In working with this apparatus, such a quantity of lead must be melted as corresponds to the capacity of the crystallizing vessel, and as soon as it is fused it is run into the crystallizer, and the operation of stirring commences.

The formation of crystals is effected in the usual way by gradually lowering the temperature, and as soon as the required amount has been formed, the lateral valves are opened and the liquid alloy run into the receiving-pots. Lead, containing the same amount of silver as the crystals remaining in the crystallizer, is now fused in the melting-pot in such quantity as to make up with them another charge. This is tapped at a high temperature upon the crystals, and a second quantity of enriched alloy is obtained. These operations are continued until rich alloy suited for cupellation is obtained on the one hand, and poor lead ready for the market on the other.

As soon as a sufficient amount of each class of lead has been accumulated to make up a full charge, with the crystals remaining in the crystallizer, the process may be continued with unbroken regularity. It is stated that by this method the cost of labour is only about one-half of that by the ordinary process, and that, in addition, a considerable saving of fuel is effected.

At the lead-works of MM. Luce & Rozan, near Marseilles, a process of crystallizing by steam has been introduced. The apparatus employed is similar in form and arrangement to that employed by M. Laveissière at Rouen; but instead of using machinery for stirring the lead, the same object is more simply and effectually accomplished by introducing a jet of high-pressure steam into the molten metal. The agitation caused by the ascent of the steam through the mass of lead is very great, and necessitates that the sides of the pot should be higher than usual above the surface of the metal, and also that it should be provided with a strong iron cover, having segmental openings fitted with hinged flaps, which the workmen open, one after the other, as required. To this cover is fitted a large iron pipe, through which the escaping steam and dust are carried into condensing chambers communicating with the chimney. The dry steam is maintained at a uniform pressure of about 45 lbs. per square inch; experience having shown that a lower one is insufficient to overcome the resistance offered by the mass of crystals, and that steam at a higher pressure acts too energetically as an oxidizing agent. Before turning on steam, care must be taken to let out any condensed water that may have collected in the steam-pipes, since otherwise an explosion would ensue.

The steam crystallizing process has been adopted, among other localities, at Newcastle, Stanhope-in-Weardale, Eureka Nevada, and Przibram.[1] At the latter place, the crystallizer is a flat-bottomed cylindrical pot, 4½ feet both in diameter and depth, holding 20 tons. Two trough-shaped melting-pans, each of 7 tons capacity, are placed behind and a little above the top of the crystallizer. They are loose on their seats, so that the melted lead may be run out by tipping them at one end with a crane.

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[1] Balling, Metallhüttenkunde, p. 292.

Steam is admitted by a horizontal pipe a little above the bottom of the crystallizer, having an iron baffle plate fixed above it in order to distribute the current through the molten lead. The conical cover of the crystallizer has two working openings and is jacketed; the flue gases of the melting fire-place being passed through the jacket whenever it is desired to clear the inner cover from splashed lead solidified upon it. Each operation lasts about four hours, of which three are required for melting and one for crystallizing. During the latter stage steam is passed through the metal until the resistance is so great that there is no longer any perceptible boiling. The enriched lead is run off by a spout closed by a slide valve, which is heated by a special fire to prevent it from becoming choked by solid lead. The crystals are kept back by a perforated straining plate. The lead, which is liquated but not softened before crystallizing, contains about 150 ounces of silver, and is divided by ten or twelve operations into rich and poor portions, with 350 to 450 ounces, and 0·3 to 1 ounce per ton respectively. The latter is passed through the softening-furnace to remove the last traces of antimony before it is fit for sale. From six to seven charges are usually worked in twenty-four hours, with a consumption of 4 cwts. of coal per ton of lead treated. As compared with the ordinary Pattinson process, the saving in fuel is about 40 per cent., and in wages 20 per cent., but the production of dross and oxides is larger, and more frequent repairs are required. The crystallizers require renewal after 120 days and the pans after 40 days' working.

DESILVERIZATION BY ZINC. PARKES'S PROCESS.—When lead and zinc are melted together, and the fused mixture is allowed to cool slowly, the zinc solidifies first, forming a layer on the surface of the metallic bath, which may be readily removed in the form of a crust containing nearly the whole of the silver originally present in the lead. Patents for the desilverization of lead by this means were granted to Mr. Alexander Parkes, of Birmingham, in the years 1850, 1851, and 1852; in 1859 this process was in operation at the works of Messrs. Sims, Willyams, Nevill & Co., of Llanelly. As it was there carried out, the process is conducted as follows:—A charge of 7 tons of the lead to be desilverized is fused in a large cast-iron pot, close to which is a smaller one for the fusion of the necessary zinc. As soon as the whole of the lead has become melted it is made to boil, by the insertion of a green pole, and the oxides, which rise to the surface, are removed by a perforated skimmer. The temperature of the metal is now raised to the melting-point of zinc, and zinc is added in the fused state in the proportion of about 1½ lb. for each ounce of silver contained in the lead operated on. The mixture is now well stirred during about two hours, the fire subsequently withdrawn, and the metal allowed gradually to cool. During the process of cooling, any of the zinc alloy which may adhere, in the form of rings, to the sides of the pot must be removed by means of a piece of wood, and as soon as the surface has sufficiently hardened it is collected by skimming with a perforated ladle. The alloy thus obtained is a mixture of lead and zinc containing silver, and is subjected to a process of liquation in an inclined iron retort, where it is heated somewhat above the melting-point of lead. The eliquated lead thus obtained should assay about 10 ounces of silver per ton, and the residual zinc will contain, in addition to a considerable amount of silver, about 50 per cent. of lead. This eliquated lead is allowed to accumulate until the quantity is sufficient to form a charge for the melting-pot, when it is fused and skimmed in the usual way, but without addition of zinc, as the proportion of that metal present is sufficient for the removal of the silver. The zinc, after being as far as possible freed from lead by liquation, is distilled in a Belgian furnace, in admixture with lime and coal-dust; the residue in the retorts consists of lead and pulverulent matter. The former is re-melted, skimmed, and cupelled; and the latter added to the charges of an ordinary lead furnace.

The lead which remains in the melting-pot, after the removal of the argentiferous alloy from its surface, contains a certain amount of zinc, which is removed by treatment in the ordinary softening furnace. The furnace used for this purpose may be of the usual dimensions, and the melted lead is maintained at a full red heat for a period varying with its quality. A calcination of from nine to twelve hours will generally be found sufficient, but samples must be taken from time to time for the purpose of testing the progress of the operation. The lead is skimmed twice; once about three hours after charging, and a second time shortly before tapping.

After a comparatively short trial, Parkes's process was abandoned at the Llanelly works, as practical difficulties were experienced which could not at the time be overcome. In 1851 this subject was carefully investigated by Karsten and Lange at Friedrichshütte, near Tarnowitz; but the process was then abandoned on the following grounds. First, that it was difficult to so completely separate the zinc from the desilverized lead as to render it easily marketable; secondly, that the silver could not be extracted from the zinc alloy without considerable loss; and thirdly, that the separation of zinc from the lead was somewhat difficult to accomplish.

MODIFICATIONS OF PARKES'S PROCESS.—The desilverization of lead by zinc was again taken up in Germany in 1866, since which date the process has been in operation at the works of Messrs. Pirath & Co., of Commern, and at those of Herbst & Co., near Call, &c., where the process is conducted as follows:—The lead is melted in a large iron pot, and sufficiently heated to fuse a piece of zinc when placed upon its surface. The zinc is added in three successive portions; first, two-thirds of the quantity required, then one-fourth, and lastly one-twelfth. After addition of the first portion, the two metals are intimately mixed by stirring with a perforated ladle for half an hour; during this period the temperature is maintained, and at the expiration of that time the fire is damped with wet fuel, and the pot allowed to cool. As soon as the crust of zinc which accumulates on the surface has become sufficiently solidified it is removed, and any portions that may adhere to the sides are carefully detached; the skimming being continued until the lead begins to crystallize and to set on the sides of the pot. The lead is now again heated to the melting-point of zinc, the second portion of that metal is added, and the stirring and skimming conducted as before. Finally, the third addition of zinc is made, and the contents of the pot are again stirred and skimmed. On account of the richness in silver of the zinc which comes to the surface, it becomes necessary that in this case the skimming should be performed with more than usual care. The proportion of zinc added is regulated in accordance with the amount of silver present in the lead.

For the complete desilverization of argentiferous lead the following proportions of zinc have, in practice, been found necessary:—

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Lead containing | 250 | grammes of silver | per 1,000 kilos. | requires | $1\frac{1}{4}$ | per cent. | of zinc. |
| „ | 500 | „ | „ | „ | $1\frac{1}{3}$ | „ | „ |
| „ | 1,000 | „ | „ | „ | $1\frac{1}{2}$ | „ | „ |
| „ | 1,500 | „ | „ | „ | $1\frac{2}{3}$ | „ | „ |
| „ | 3,000 | „ | „ | „ | 2 | „ | „ |
| „ | 4,000 | „ | „ | „ | 2 | „ | „ |

It will be observed that the quantity of zinc necessary is by no means proportionate to the amount of silver contained in the lead. No reason can be assigned for this, but the accuracy of the figures given has been confirmed by the results of a series of trials made at Clausthal.

The argentiferous zinc removed from the mixing-pot retains a considerable amount of lead, which was at first partially separated by liquation by means of two iron pots, one placed at a higher level than the other. To the bottom of the upper pot is cast a pipe, which can be opened or closed as required. The zinc skimmings are strongly heated in this pot, and the eliquated metal which collects in the bottom is tapped into the lower vessel, while the argentiferous residue remains in the upper one in the form of a pulverulent mass. The eliquated metal carries with it a little silver and zinc, and after slowly cooling, it is skimmed, the skimmings being again subjected to liquation. The residual lead, which is now poor in silver, is added to the original metal previously to the introduction of the third portion of zinc. The argentiferous zinc residues are finally melted in a small blast-furnace with an admixture of lead slags and tap-cinder, and the lead obtained cupelled in an English refining-furnace.

At the works of Herbst & Co. the dezincification of the desilverized lead has been effected by the use of chloride of lead. For this purpose the poor lead from which the zinc is to be removed is kept melted at a moderate temperature, for about twenty-four hours, under a layer of chloride of lead, when, by frequent stirring, the zinc is converted into zinc chloride with the separation of metallic lead. The lead chloride used is prepared by treating fume from the flues with hydrochloric acid, and is consequently not pure.

According to Illig the dezincification of desilverized lead may be effected by passing it through a small blast-furnace with the addition of sand and tap-cinder. There can be no doubt but that the zinc may be thus removed, although a considerable loss of lead must ensue, and the lead so treated would be more or less hardened.

In the year 1866, Clemens Fleming Flach, "of Call, in the kingdom of Prussia," obtained letters patent in this country for "Improvements in Extracting Silver from Lead." By the system described in the specification of this patent the lead is first desilverized by two or more successive additions of zinc in the usual way, and the resulting argentiferous alloy is melted in a blast-furnace with siliceous slags. The zinc is removed from the desilverized lead by fusion with slags in a blast-furnace, and by boiling the lead produced, when in a state of fusion, by the introduction of green poles. The process for separating zinc from the desilverized lead in the blast-furnace has not, however, been generally adopted.

This process in its modified form is simple, and is conducted in the following way. At a height of about 8 feet from the level of the floor three cast-iron pots are set in brick-work over separate fire-places; the largest of these pots is of a capacity to contain a charge of about 20 tons of lead, while the other two are much smaller, each holding about 6 tons of metal.

The lead to be desilverized is melted in the larger pot, where the usual quantity of zinc is added, and the argentiferous alloy removed, in perforated ladles. This is deposited in one of the smaller pots, in which a portion of the associated lead is separated by liquation; this collects at the bottom, and the concentrated argentiferous alloy is skimmed from its surface by the aid of a perforated ladle. When one of the smaller pots has become filled with skimmings from the large one it is subjected to liquation; the other in the meantime serving for the reception of skimmings from the larger pot.

The argentiferous alloy, from which as much as possible of the lead has been previously separated, may be smelted with an admixture of lead slag and tap-cinder, in a low blast-furnace, blown by three tuyers. The lead thus obtained is finally subjected to cupellation. During the process of smelting in the blast-furnace the zinc becomes volatilized, and is carried off by the flue in the form of zinc oxide; the draught is accelerated by the introduction of a steam-jet into the flue leading from the top of the blast-furnace to the chimney.

The lead eliquated in the smaller pots, from the skimmings removed from the large one, is added to the next charge of original lead. That remaining in the large pot, after the removal of the zinciferous crust, is tapped into the pan of an improving-furnace, situated at a lower level, where it is kept at a red heat during about twelve hours, and is occasionally skimmed; at the expiration of this time it is drawn off into a cast-iron pot and laded into moulds as market-lead.

At the Par Smelting Works, where this system was in operation for several years, a saving of 40 per cent. was effected over the cost of desilverization by the Pattinson process, and the excess of silver was larger than was obtained by that method; the loss of lead was stated to be slightly over 2 per cent. Guillem & Co., of Marseilles, state that by this system their rich lead is concentrated so as to contain 9 per cent. of silver, and they estimate the saving in cost, as compared with the ordinary Pattinson process, at 45 per cent.

By Cordurié's process, for which a patent was obtained in this country a few weeks prior to the date of Flach's, the dezincification of desilverized lead is effected by the agency of steam. Superheated steam is passed through the desilverized lead, heated to redness, until hydrogen gas ceases to be evolved. By this means the zinc is oxidized by the oxygen of the steam with an equivalent evolution of hydrogen, while the lead is but slightly attacked. The zinc oxide which rises to the surface is subsequently skimmed off. The argentiferous crust of zinciferous alloy is also exposed to the action either of hot air or of superheated steam; the zinc is thus oxidized, together with a certain quantity of lead, and the mixed oxides are separated from the residual argentiferous lead either by liquation or by skimming. The rich lead resulting from this treatment is cupelled.

Schnabel has introduced a method of treatment for the rich argentiferous oxides produced in this process, which contain both zinc and lead oxides and metallic lead, and are exceedingly refractory. This consists in digesting the oxides with a hot ammoniacal solution of carbonate of ammonium in gas-tight vessels under pressure, when zinc oxide dissolves, leaving the silver lead and lead oxide in a form suitable for addition to the refinery test. The ammoniacal solution, after passing over zinc plates to remove copper, is distilled to recover the ammonia, and the residue of basic zinc carbonate is converted by calcination into zinc oxide, which is used as paint.

In America the larger portion of the lead smelted in Nevada, Utah, and Colorado is either sent to the Pacific, the Missouri Valley, or to the Eastern States to be softened and desilverized. The largest and most important of these establishments are Balbach's Smelting and Refining Works at Newark, New Jersey, and the Omaha Refinery in Nebraska. The lead treated is usually very impure, and requires a preliminary calcination in the softening-furnace in order to remove antimony and other oxidizable metals. The softened lead, containing from 0·3 to 1 per cent. of silver, is run into pots, where it is treated with zinc, the quantity of the latter metal being so proportioned that the skimmings, after distillation, may leave a residual rich lead containing from 8 to 10 per cent. of silver. On account of the highly argentiferous character of the lead operated on, the proportion of zinc used is unusually large, amounting to from 1½ to 3 per cent. of the lead desilverized.

The zinc alloy is first chilled by water, and subsequently removed in the form of a coherent crust from the surface of the lead bath, after which it is subjected to liquation in a reverberatory furnace.

This process is conducted at so low a temperature that the adherent lead only is melted, while the less fusible zinc and silver alloy is merely softened. During this operation care is taken to exclude air as completely as possible, in order to prevent the formation of metallic oxides. The lead thus separated, which is nearly free from silver, runs down the inclined bed of the furnace into the desilverizing pots, while the argentiferous alloy, containing from 4 to 6 per cent. of silver, is chiefly in the metallic state; a point of much importance in facilitating its subsequent treatment by distillation.

The furnace used for the latter operation, the joint invention of Messrs. Balbach and Faber du Faur, is an air-furnace mounted on trunnions, and turning by the action of worm-wheel gearing.

An open arch turned over the fire-place supports a single bottle-shaped retort, placed with the mouth inclining upwards, and the neck projecting beyond the front wall. The retorts made of black-lead, fire-clay, and burnt fire-bricks, are 2 feet 7 inches in length, and have a thickness of 2¾ inches. The charge, consisting of from 220 to 330 lbs. of eliquated zinc skimmings, broken up on an iron plate in front of the reverberatory furnace, is, while still hot, introduced into the retort, also at a red heat, together with from 4½ to 9 lbs. of charcoal dust, and is fired for some eight to ten hours at the full heat of the furnace.

Nearly 8 cwts. of coke are consumed during this operation, and the zinc separated is condensed in a receiver usually formed out of the neck of an old retort. In this way about 40 per cent. of the zinc is collected in the metallic state, together with about 20 per cent. in the form of dust and oxide, both being practically free from silver. The residual rich lead, containing only traces of zinc, is removed by turning the furnace on its trunnions until the neck of the retort is below the horizontal, when it flows out into a mould placed for its reception, and is then ready for refining in the ordinary test-furnace. A retort is usually capable of working from fifteen to thirty charges without renewal. By taking care to prevent oxidation of the zinc during the operation of liquation the expense of dealing with large quantities of argentiferous zinc oxide is entirely avoided.

This process, which is considerably more economical than the fusion of the argentiferous alloy in a blast-furnace, is now generally adopted in the United States under the name of the Balbach process, and has also been introduced into the principal lead refineries on the Continent of Europe and in this country. The cost of desilverizing lead by zinc is not only less than by any of the processes by crystallization, but the loss of silver in the market-lead is also thereby almost entirely avoided.

## CUPELLATION OR REFINING.

ENGLISH REFINERY.—In this country the separation of silver from lead is conducted on a hearth made of bone-ash, forming the movable bottom of a reverberatory furnace. The hearth or *test* is contained in an elliptical iron framing, seldom less than 5½ or 6 inches in depth, usually about 4 feet in its greater and 3 feet in its lesser diameter. To support and strengthen the bottom of the test, this frame is provided with four parallel cross-bars, 4½ inches wide, and, like the ring itself, half an inch in thickness. There are also two bars, called "strap-bars," connecting the first transverse bar at the wider end with the ring. This framing, or *test-ring*, is most frequently made of wrought-iron, the cross-bars being attached by rivets, but in some cases it is formed of cast-iron, and is then, including the bars across the bottom, cast in one piece.

To prepare a test, the frame is filled with bone-ash well beaten in layers, after having been previously moistened with a little water, holding a small quantity of pearl-ash in solution; the presence of a minute proportion of this substance has the effect of giving consistency to the cupel when heated. After the framing has, in this way, been filled with slightly moistened bone-ash, solidly beaten down, a cavity is carefully scooped in its upper surface, until the sides are left 2 inches in width at top, measuring from the iron ring, and gradually widening to about 3 inches at bottom; the thickness of the bottom itself may be about 1½ inch.

At the front, or wider end of the test, three holes are usually bored through it; of these the central one is made to communicate, by means of a channel, with the fluid litharge in the annular cavity, formed between the test and the slightly curved edge of the metallic bath. This allows the fused oxide of lead to escape as rapidly as it is produced, and when it becomes so much corroded by the action of litharge as to be no longer serviceable, it is closed by a little moistened bone-ash, and a new channel is opened in connection with one of the other holes.

The test thus prepared must be kept for some time in a warm place, to become thoroughly dry, and may be then placed in the refining furnace, of which it forms the bottom. Figs. 196, 197, 198, represent, respectively, an elevation, a horizontal section, and a vertical section, through the longer axis, of the refinery employed at the Couëron Lead-Works.

The size of the fire-place, A, varies with the other dimensions of the furnace, but it is usually nearly square, and may measure about 2 feet by 2 feet 4 inches. This is separated from the body of the furnace by a bridge, from 14 to 18 inches in width, so that the products of combustion pass from it directly over the surface of the test, and escape to the main flue by two separate apertures, *a*. The test, B, is maintained in its position, so as to form the furnace bottom, by being tightly jammed, by means of the wedges, *b*, of which there are four, supported by two iron bars, against the iron ring, *c* (fig. 198), firmly built into the masonry of the furnace. The application of heat to the test must, at first, be regulated with care, since if the temperature were too abruptly raised it would be liable to crack and exfoliate. As soon as it has become properly annealed it is heated to redness, and a charge of the lead to be operated on is introduced. In the majority of cases this is previously fused in an iron pot, C, set over a fire-place, and provided with a gutter, *d*, through which the molten lead is laded into the cavity of the test. When first introduced into the furnace, the liquid metal becomes covered by a greyish dross; but as soon as it has acquired the full temperature of the test, the surface of

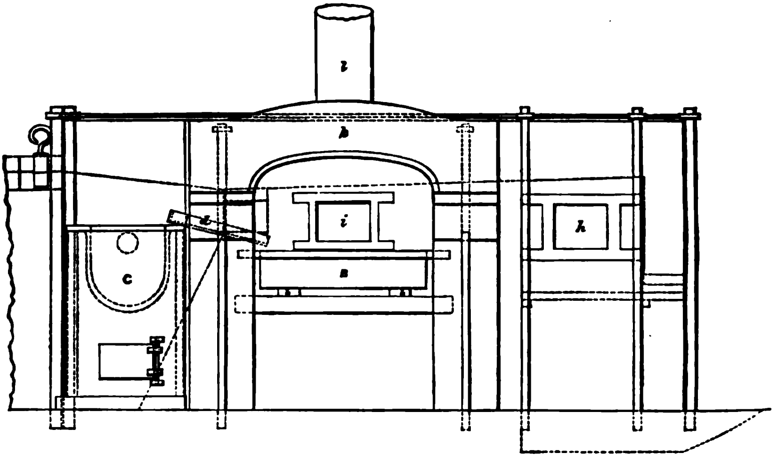

Fig. 196.—Refinery; front elevation.

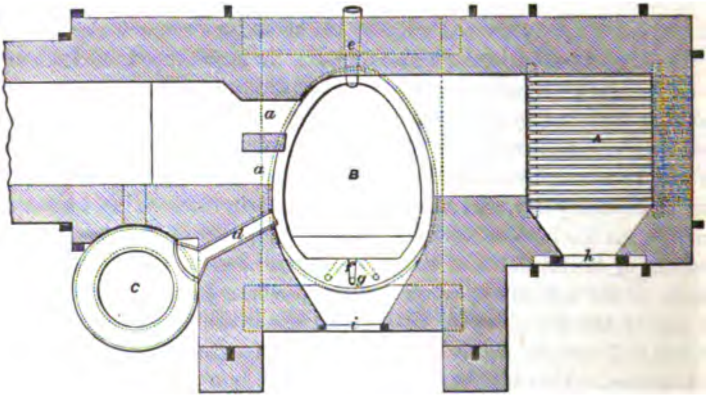

Fig. 197.—Refinery; horizontal section.

the bath uncovers, and fused litharge begins to make its appearance. The blast is now turned on through the nozzle, *e*, and the melted litharge is thus driven from the back of the test up towards the breast, whence it escapes by a channel or *gate*, *f* (fig. 197), in connection with the central aperture, *g*, through which it falls into a shallow cast-iron pot, mounted on wheels, and furnished with a long handle. When the channel, *f*, has become so much acted upon by the litharge as to be no longer serviceable, it is closed and a new one made, as shown by the dotted lines, communicating with one of the holes situated to the right or left of the longer axis of the test. Fuel is supplied to the grate through the door, *h*, while that marked *i* is used for the purpose of watching and regulating the operation; the fumes are carried off by the hood, *k*, and the iron chimney, *l*. Sometimes, instead of feeding the test with rich lead in a fused state, the metal is introduced in the form of pigs. In such cases the furnace is provided with one or more iron-lined openings, called *pig-holes*, through which the metal is introduced at the back of the furnace, in the vicinity of the blast-pipe. The blast, which is usually supplied by a fan, not only furnishes the oxygen necessary for the formation of litharge, but also sweeps the fused oxide along the surface of the metal towards the breast. In some cases, a blast, produced by means of a jet of steam, is employed, instead of a current of air from ordinary blowing machinery. In proportion as the surface of the metal in the test becomes depressed, through constant oxidation, and the continual removal of the resulting litharge, additional lead is supplied, either from the melting-pot, C, or in the form of pigs, so as to keep it nearly at the original level. In this way the operation is continued until the lead has become so much enriched as to render it desirable that it should be tapped. When the lead operated on contains about 600 ounces of silver per ton, this operation is generally performed at intervals of eight hours; during this time about 32 cwts. of lead will have been introduced, and from 4 to 6 cwts.

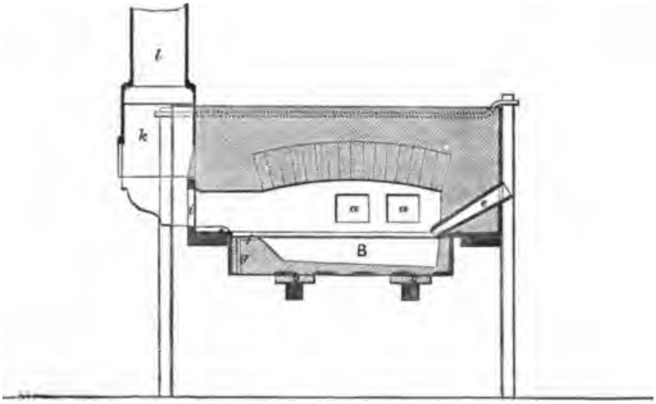

Fig. 198.—Refinery; transverse section through tuyer.

of enriched alloy will remain in the bottom of the test. The removal of the highly concentrated argentiferous lead is generally effected by making a hole in the bottom of the test with a drill contrived for that purpose, and running it off into a cast-iron pot, on wheels, placed under the furnace for its reception; in some cases, lading out the rich lead is resorted to instead of tapping. When the concentrated rich lead has been thus drawn off, the tapping-hole is closed by a pellet of bone-ash, kept in its place by an iron plate, and another charge is immediately introduced. The reason for thus removing the enriched argentiferous lead from the test is to avoid the distribution of too large an amount of silver in the litharge, which would be the case if fresh lead were continuously added to a constantly increasing accumulation of silver. In works where the concentration of silver is effected by the use of zinc, it is not customary to tap the highly enriched lead from the test; since it is found more advantageous to carry on the operation without interruption, and to add the metal reduced from the final rich litharge to the original lead, before the introduction of zinc. When tapping or lading out is resorted to, the whole of the lead operated on is thus further enriched, and the resulting highly argentiferous alloy is finally subjected to cupellation, either on the same test, or in another specially prepared for the purpose.

The appearance of the surface indicates the precise period at which the operation is terminated; the blast is then turned off, and the fire removed from the grate. The *plate* of silver is thus allowed to set, and as soon as it has done so, the wedges, *b*, are removed from beneath the test-frame, which, together with its contents, is lowered upon a small iron bogie-waggon, and taken away to cool. The silver is subsequently detached from the test, and any adhering particles of litharge, slag, or bone-ash are removed by scraping with a wire brush.

An ordinary refinery works off from 4 to 5 cwts. of lead per hour, and consumes from 6 to 7 cwts. of coal per ton of lead oxidized. The plate obtained may vary in weight from 5,000 to 10,000 ounces, and usually contains from 997 to 998 parts of silver in a thousand.

The loss of lead experienced during the operation of refining is about 7 per cent. of the weight worked; the process is conducted by one refiner on each shift, occasionally assisted by a labourer. The test bottoms, which are saturated with litharge, and contain a certain amount of silver, are broken up and smelted, either in the blast-furnace or otherwise. The yield of silver from lead by Pattinsonizing and subsequent refining on the test is usually about 2 per cent. above that indicated by assay of the ores. This is due to subsequent recovery of silver from fume, litharge, &c., which is lost in assaying. Strictly speaking, refining is not cupellation but scorification, as the bulk of the litharge flows like slag instead of being absorbed by the test, while in cupellation complete absorption of the whole of the litharge is essential to the success of the operation.

## GERMAN CUPELLATION (ABTREIBEN).

The German refinery or *Treibheerd*, which is generally used in Continental establishments, is represented in an old form in figs. 199, 200, of which the first is an elevation, and the second a horizontal section.

This is a kind of reverberatory furnace, having an oval hearth and a lateral fire-place. The foundation of the hearth, A (fig. 200), is composed of fire-bricks closely set on edge upon a solid stratum of firmly compressed slag, and is covered by the marl refining bottom (*Mergelheerd*), which corresponds to the bone-ash test of the English refinery. Formerly this was made of natural marl finely-powdered; but now a mixture of 3 or 4 parts of limestone or dolomite with 1 of clay is generally used. The roof is a dome of iron internally plastered over with clay, and capable of being either removed or lifted into its place by chains attached to a lever supported by the movable crane, C. In the sides of this furnace are five openings; by the largest of these, *d*, the flame passes from the fire-place, B, into the interior of the hearth; the two openings, *l*, serve for the introduction of the tuyers, by which a blast is thrown on the fused metal, for the purpose of causing its oxidation, and, at the same time, forcing the litharge formed on its surface towards the aperture, E, from which it escapes in a fused state; finally, F is the opening, through which a portion of the lead to be operated on is inserted, in the form of lenticular discs. At the commencement of the operation the opening, E, is partially closed by the breast of the hearth, but in proportion as the operation advances channels or gateways are succes-

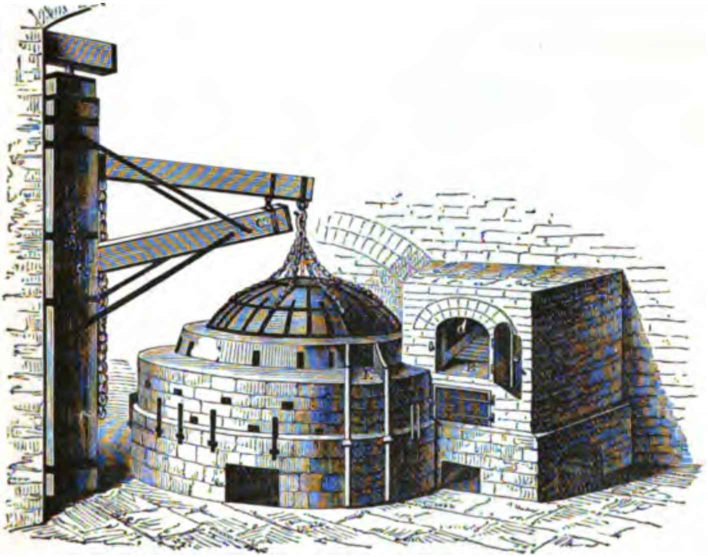

Fig. 199.—German Cupelling Furnace; elevation.

sively cut down by a serrated iron bar to the level of the litharge contained in the furnace. The litharge which escapes from this opening either flows down to the floor of the building, or is collected in a mould placed in front of E.

In a new furnace the whole hearth above the brick bottom, A, is filled with marl and beaten down by heated rammers, resembling broad-toothed rakes, to the required concave form of the refining bed, which is deepened about 12 or 14 inches in the middle, and has a circular hollow about an inch deep (*Blickspur*) cut in the lowest part for the final collection of the silver; but when the furnace is at work, only the upper saturated crust, about 3 or 4 inches thick, is renewed after each operation. In either case the dome is removed by the crane before making the new bed.

When working with unsoftened lead, about 5 tons are usually treated at one operation, and of this a little less than three-fourths is introduced into the furnace before lighting up; the remainder is added at successive intervals during the progress of the refining. As soon as the marl bed has been dried, about 75 cwts. of lead are charged, in the form of small hemispherical pigs, which are placed with their convex surfaces downwards, so as not to injure the bottom. On the centre of the heap of lead thus formed are placed some billets of wood; these are ignited by means of a shovelful of burning charcoal, and the movable cover, after being carefully dropped into its place, is luted round with fire-clay. The blast is then turned on, and a fire of billets is made upon the grate. From three to five hours are required for the complete fusion of the mass, and when this has been accomplished the surface of the molten lead is found to be covered by a scum of refractory oxides known as *Abzug*. In order to facilitate the removal of this, by skimming, the temperature is now raised, and it is drawn through the litharge channel, E. This skimming

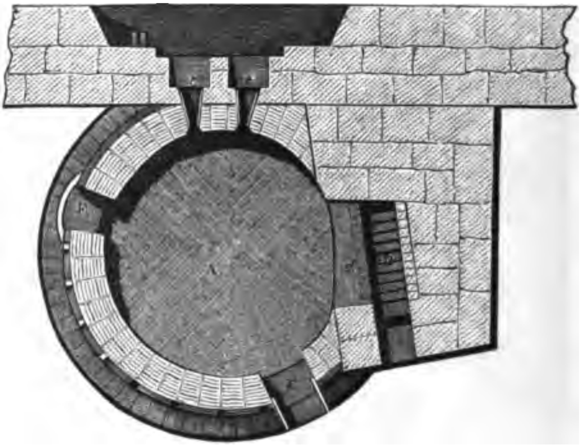

Fig. 200.—German Cupelling Furnace; horizontal section.

occupies about an hour, during which time fresh quantities of scum are continually rising to the surface. The bath of lead gradually acquires a gentle circular movement, and becomes bright and clear, but is quickly obscured by a covering of impure litharge, or *Abstrich*.[1] This, which is removed through the litharge channel, ordinarily begins to flow about an hour and a half after the skimming off of the last Abzug, and continues to escape during about the same length of time, after which pure litharge makes its appearance. Litharge subsequently continues to flow from the furnace until the *Blick*, or brightening of the residual silver, takes place; this generally occurs in from thirty to thirty-three hours after first turning on the blast.

As soon as the flow of Abstrich has ceased, the remaining lead is introduced, two pigs at a time, through the opening, F, which, besides being used for this purpose, serves as a passage for the escape of a large portion of the products of combustion. The lead thus added is placed on a part of the bottom, which is raised for that purpose slightly above the ordinary level. In thus adding the second portion of the lead to be cupelled, a hard refractory mass of Abzug is left behind on the part of the bottom where the fusion of the pig is effected; this is from time to time loosened and removed.

Towards the close of the operation the temperature requires to be considerably increased for the purpose of keeping the alloy, which is then rich in silver, in a sufficiently liquid state. The nozzles of the tuyers supplying air to the blast are sometimes covered by small valves called "butterflies," which, being hung before them, serve to spread the blast over the surface of the metallic bath. The operation is continued until the greater portion of the lead has been removed in the form of litharge, and a plate of nearly pure silver of the shape of the cavity in the bottom remains.

Immediately after the brightening has taken place, the workmen throw water over the surface of the metallic residue, and the *Blicksilber*, which is not pure, but contains about 5 per cent. of lead, &c. (*Schwarzblicksilber*), is removed from the furnace for the purpose of being refined.

The average loss of lead during cupellation by the German process is estimated at 8 per cent.; but when, as was formerly the case, the bed was formed of lixiviated wood-ashes mixed with a little lime, this loss is stated to have been sometimes as high as 14 per cent. According to Winkler (1837), about 250 cubic feet of cord-wood are required for the cupellation of 5 tons of argentiferous lead in the state in which it is obtained from the smelting-furnace.

Since the use of the softening furnace has become general, much larger quantities of lead are refined in one operation than was formerly the case, the furnaces being made larger, and the circular form of hearth modified to a more nearly elliptical or even rectangular shape. Thus at Freiberg the initial charge is 10 tons for small and 15 tons for large furnaces, and a further quantity of 30 to 35 tons is added to the former and 35 to 45 tons to the latter during the operation, which lasts six to seven days, until the concentration to 80 per cent. of silver is effected. Bohemian lignite is used as fuel, with a blast under the grate. At Przibram, 22½ tons of lead are charged at once on a furnace with a hearth about 11½ feet square, and refined in three days, giving a cake of 95 per cent. silver, weighing upwards of 2 cwts. The coal burnt is about 20 per cent. of the weight of the lead refined.

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[1] The terms Abzug and Abstrich are applied differently in different districts; but generally speaking the latter may be considered to be antimonial or arsenical litharge, and the former unmelted dross that must be drawn or skimmed.

Refining the Blicksilber.—This operation, which is called *Feinbrennen*, may be performed in various ways, all founded on the principle of the separation of impurities by oxidation, at a temperature somewhat above the melting-point of silver.

The most ancient process appears to be that of refining by means of a blast on an open test, of which the general arrangement is somewhat similar to that of the ordinary blacksmith's forge. This method of refining is described by Agricola in his 'De Re Metallica' (1561), and five illustrative woodcuts are given of the apparatus then employed; it essentially consists of a large cupel beaten into an iron dish, and of double bellows for supplying a constant blast. The test was formerly made of a mixture of two-thirds lixiviated wood-ashes and one-third bone-ash; subsequently a mixture of bone-ash and sulphate of barium was made use of, but latterly marl similar to that used for the furnace bottom was employed. A hollow was cut with a curved knife in the centre of this test, which was placed in a cavity prepared for its reception in the top of the hearth.

The Blicksilber was cut into pieces, and piled upon the test, which was surrounded by a sheet-iron hoop filled with charcoal; ignited charcoal was placed before the nozzle, and the blast was turned on. The fusion of the silver was usually complete in the course of about half an hour, when the iron hoop was taken away, and the charcoal removed from the surface of the metal. Small billets of dry wood were now laid before the tuyer; these were replaced by fresh ones as fast as they were consumed, and care was taken to remove any ash that might fall upon the surface of the metallic bath. During this operation the silver was occasionally stirred with an iron rod, and as soon as the drop of metal which adhered to its extremity was observed to *vegetate* on cooling, the blast was stopped, the fire removed, and the cake of fine silver cooled with water until it had solidified. It was then removed from the test, and after being cleared from adhering particles of slag and litharge, was ready for the market.

Instead of refining on an open test, the operation was sometimes conducted under a muffle into which a blast was admitted; the refining of Blicksilber is now usually conducted either on a movable test, like that employed in the English process, or on a fixed marl bottom in a reverberatory furnace without a blast. From 16 to 22 cwts. of crude silver are brought up to 996–998 thousandths fine in twelve hours.

### TREATMENT OF OXIDIZED REFINERY PRODUCTS.

The oxidized products, dross, litharge of all kinds, and saturated bottoms, whose total weight is about 10 to 20 per cent. more than that of the lead operated upon, are either *revived*, *i.e.*, reduced to the metallic state, returned to the ore-furnaces, or sold as oxide of lead, according to their condition of purity and greater or less richness in silver. The earlier products, Abzug and Abstrich, are, like the analogous dross of the calcining-furnace, converted into hard lead. Litharge, when allowed to cool quickly in the air, collects in stalactitic lumps of a greenish yellow colour, which are compact and somewhat tough; but when it is moulded into large blocks, the interior portions, which remain fluid for some time after the outside has become hardened, are converted into a slightly coherent mass of crystalline scales of an orange-red colour. This, known as red-flake- or market-litharge, is the form of oxide of lead generally used in chemical and other manufactories, and, when possible, is sold as such, as its pulverulent form renders it less suitable for reduction than the harder variety, which is called yellow-lump- or reviving-litharge.

REDUCING.—The reduction to the metallic state of litharge from the refinery, the pot-dross, and the dross from the calcining pans, is, in this country, effected in a reverberatory furnace, somewhat resembling in form that used for smelting; excepting that its dimensions are smaller, and that the sole, instead of being lower beneath the middle door than at any other part, gradually slopes from the fire-bridge to the flue at the opposite extremity. Here there is a depression in which is the tap-hole; this constantly remains partially open, and from it the reduced metal continually flows into a small iron pot, placed on the side of the furnace. Under this pot a fire is lighted, and the lead is subsequently laded from the pot into moulds.

Before being thrown into the furnace, the litharge is mixed with small coal, and is then charged on that part of the hearth which lies before the fire-bridge. To prevent the fused oxide from attacking the bottom of the furnace, and also to afford a sort of hollow filter for the liquid metal, the workman, before charging the oxide to be reduced, covers the hearth with a layer, about 2 inches in thickness, of bituminous coal. The heat of the furnace soon causes the ignition of this stratum, and it quickly becomes burnt to the state of a spongy cinder, upon which the mixture of litharge and carbonaceous matter is charged. The small coal in the charge causes the reduction of the litharge, which, assuming the metallic form, flows gradually through the interstices in the cinder, and falls into the depression at the extremity of the hearth; whence it gradually flows through an iron spout into the external pot in which it is collected. The surface of the charge is, during its elaboration, frequently stirred with an iron rake, for the double purpose of exposing new surfaces to the action of the furnace, and also to allow the reduced lead to escape more readily.

Additional quantities of the material operated on, mixed with coal, are from time to time charged into the furnace; at the termination of the shift, which commonly extends over twelve hours, the tap-hole is opened, and, after the escape of the whole of the lead, the residual *lead-cinder* is withdrawn. A new floor of cinders is then formed, and the operation continued as before. A furnace with a bottom 8 feet in length and 7 feet in width will reduce 5½ tons of lead from litharge in the course of twenty-four hours. About 3 cwts. of coal are required for the reduction of each ton of litharge. No fresh material is charged for a considerable time previous to the termination of a shift, and the lead-cinder then withdrawn is, in the majority of cases, smelted in the slag-hearth or blast-furnace.

In Germany, litharge-reviving is usually done with coke or charcoal in a low blast-furnace, about 10 per cent. of slags from previous reducing operations or from ore-smelting being added, and sometimes a little fluor-spar. In Freiberg the 8-tuyer ore-smelting furnaces are used, and reduce about 60 tons of litharge daily, with an expenditure of 3 to 4 tons of coke. The lead, when necessary, is softened and desilverized. The rich litharge produced in the final stage of refining, when the lead is concentrated to 2,000 ounces of silver and upwards, is separated and treated apart, as it gives lead with 70 to 100 ounces of silver. Saturated tests are reduced with fluor-spar flux in a slag hearth; but the marl bottom of the German refinery is usually returned to the ore-smelting furnaces.

*Electrolytic Lead-Refining.*—Keith's process for the production of pure lead by electrolysis has been for some years in use in New York. The furnace-lead, which need not be softened, is cast into thin plates about 4 feet long, weighing 45 lbs., which are enclosed in muslin bags and made the anodes in an electrolyzing cell containing a solution of sulphate of lead in acetate of sodium; the cathodes, plates of pure lead, are placed about 2 inches from the anodes. The bath is heated by steam to about 38° C., and the current of a Weston dynamo, making 1,500 revolutions per minute, is passed through the cell. The anodes are corroded and dissolved with an equivalent deposition of lead on the cathodes in a loose crystalline powder, which falls to the bottom of the cell, while the gold, silver, and other insoluble matters are retained in the muslin bags, and collected for further treatment when only 2 or 3 per cent. of the lead plate remains undissolved. With a system of forty-eight decomposing cells, each containing 50 anodes, 10 tons of 180-ounce silver lead, with 2¼ per cent. of antimony and arsenic, are refined daily by a 12-horse power steam-engine. The precipitated lead contains 11 grains of silver per ton, and must be compacted by hydraulic pressure before it can be melted.

### LEAD FUME.

Lead being to a considerable extent volatile at high temperatures, a notable loss of that metal is experienced during the various operations of smelting, refining, reducing, &c.; different means are consequently employed for the purpose of collecting these fumes and for rendering them available as a source of lead. The most efficacious method of collecting the lead carried off in the state of fume is by the use of long flues of considerable transverse area. Numerous other contrivances, such as drawing the fumes through water, passing them through condensing chambers, the introduction of water in the form of spray, blowing steam into the flues, &c., are sometimes resorted to, with more or less satisfactory results. The following are the respective lengths of the flues at the smelting-works belonging to Mr. Beaumont, as furnished by Mr. Sopwith to Dr. Percy:—

| | | Yards. |
|---|---|---|
| At Allen Smelt Mill | one | 4,451 |
| | the other | 4,338 |
| At Allenheads Mill | | 3,424 |
| At Rookhope Mill | | 2,548 |
| | | 14,761 |

The total length of the above flues is consequently about 8½ miles; their transverse area is not uniform, but their average height is 8 feet and their width 6 feet. In one year as much as 800 tons of lead have been extracted from the fumes obtained from these flues. At Keld Head 96 tons 13 cwts. of lead were obtained from fume resulting from the production of 1,374 tons, or in the ratio of 7·03 per cent. At Pontgibaud, where the flues are 500 metres in length, and the ores are smelted in blast-furnaces, 3·67 per cent. of the lead is obtained from fume. At the Wildberg Smelting Works, Germany, where the flues were 800 feet in length, and smelting was conducted in the Castilian furnace, 1½ per cent. of the assay produce of lead was obtained from fume.

At Ems, in Nassau, a flue of varying section about 1850 yards, or including branches 2260 yards, long is used. The wall surface, of about 180,000 square feet, is increased by more than 120 per cent. by thin sheet-iron plates hung parallel to the direction of the current on the cooler sections near the chimney, which is 730 feet above the furnaces. The dust collected from smelting 14,600 tons of ore with 6,248 tons of lead, yielded 500 tons of the metal, or about 8 per cent. of that contained in the ore.

The lead contained in fume, to a large extent, exists in the form of sulphate, and is recovered by roasting and smelting it either alone or in admixture with lead ores. Fume-lead is considerably poorer in silver than that derived directly from the ores from which it was produced.

The following are the results obtained by a series of assays of the lead-fume at Wildberg, where the average assay for silver of the lead produced was about 21½ ounces per ton:—

Samples taken from the Top of Main Flue at Distances of 100 Feet apart.

| | Lead. | Silver. | | |
|---|---|---|---|---|
| | Per Cent. | Oz. | Dwt. | Gr. |
| No. 1, near blast-furnace . . | 49 | 2 | 9 | 0 |
| ,, 2, near reverberatory furnace . | 20 | 3 | 5 | 8 |
| ,, 3, 100 feet in advance of No. 2 | 70 | 3 | 5 | 8 |
| ,, 4, 100 ,, ,, 3 | 44 | 3 | 5 | 8 |
| ,, 5, 100 ,, ,, 4 | 48 | 2 | 17 | 0 |
| ,, 6, 100 ,, ,, 5 | 47 | 3 | 5 | 8 |
| ,, 7, 100 ,, ,, 6 | 46 | 3 | 5 | 8 |
| ,, 8, 100 ,, ,, 7 | 40 | 2 | 1 | 5 |
| ,, 9, 100 ,, ,, 8 | 42 | 3 | 5 | 8 |
| ,, 10, { top of chamber at throat of refinery . . } | 49 | 22 | 17 | 8 |

Samples taken from Bottom of Main Flue at Distances of 100 Feet apart.

| | Lead. | Silver. | | |
|---|---|---|---|---|
| | Per Cent. | Oz. | Dwt. | Gr. |
| No. 1, near blast-furnace . | 53 | 2 | 9 | 0 |
| ,, 2, near reverberatory furnace . | 59 | 2 | 9 | 0 |
| ,, 3, 100 feet in advance of No. 2 | 63 | 3 | 5 | 8 |
| ,, 4, 100 ,, ,, 3 | 61 | 2 | 9 | 0 |
| ,, 5, 100 ,, ,, 4 | 49 | 2 | 17 | 0 |
| ,, 6, 100 ,, ,, 5 | 64 | 2 | 1 | 5 |
| ,, 7, 100 ,, ,, 6 | 46 | 3 | 6 | 5 |
| ,, 8, 100 ,, ,, 7 | 58 | 1 | 12 | 5 |
| ,, 9, 100 ,, ,, 8 | 62 | 2 | 17 | 0 |
| ,, 10, { bottom of chamber at foot of chimney . . } | 66 | 3 | 5 | 8 |

In the same way that, from the greater volatility of lead, the metal obtained from fume contains a less proportion of silver than that directly extracted from the ore, so also, on account of the greater oxidizability of lead, that obtained from slags is less argentiferous than that reduced from the corresponding ores.

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## SILVER.

Silver is a white metal, capable of receiving a lustre inferior only to that of polished steel. Its malleability and ductility are, next to gold, greater than those of any other metal. Pure silver is harder than gold and softer than copper; its specific gravity is 10·50; when pure it enters into fusion at a full red heat, corresponding to about 1,023° C. Fused in open vessels it absorbs oxygen in considerable quantity, sometimes amounting to 22 times the volume of the metal itself. On becoming solid, however, the whole of this gas is again expelled. This circumstance is probably, to some extent, the cause of the metallic vegetation which takes place on the surface of silver when suddenly cooled on the cupel. Heated very strongly this metal gives off metallic vapours, and between two carbon electrodes in connection with a powerful voltaic battery it is volatilized. By fusing a large quantity of silver, and afterwards allowing it to cool very gradually, cubical and octahedral crystals may be obtained on piercing the solidified crust and running off the still-liquid metal. When solutions of silver are decomposed by the action of feeble electric currents, the precipitated metal often assumes a crystalline form. Silver does not absorb oxygen at ordinary temperatures, but speedily becomes blackened on exposure to an atmosphere containing sulphuretted hydrogen, which is decomposed by it with great facility.

Heated to redness in contact with the caustic alkalies, it is not in the least affected, and for this reason is sometimes employed for making crucibles, to be used when attacking substances by caustic potash. In the presence, however, of fused alkaline silicates, silver vessels become acted on to a small extent, and the silicate is stained of a light yellow colour. Oxide of silver is reduced by heat alone, and a globule of metal is obtained.

Unless in a state of extreme division, silver is not attacked by hydrochloric acid, and even then it requires to be heated to the temperature of ebullition before decomposition of the acid is effected. By dilute sulphuric acid no effect is produced, but strong sulphuric acid, when heated, is readily decomposed, with the formation of sulphate of silver and the evolution of sulphurous anhydride. Nitric acid readily attacks silver, even at ordinary temperatures; nitric oxide is evolved, and nitrate of silver is produced. By chlorine, iodine, and bromine, silver is attacked, even in the cold.

## SILVER ORES.

Silver occurs not only in the native state, and alloyed with various other metals, but also mineralized by such elements as sulphur, selenium, chlorine, bromine, and iodine.

NATIVE SILVER; *Argent natif; Gediegen Silber.* Cubic.—Native silver is found accompanying almost all the ores of this metal, and more particularly its sulphides, sulpho-salts, and chloride. It occurs either in distinct crystals or in divergent branches, of which the extremities are often composed of minute crystals, similar to those frequently observed in specimens of native copper.

It also occurs in amorphous masses, in long filamentary strings (hair- or wire-silver), and in the shape of compressed plates. One of the largest masses of metallic silver ever obtained in Europe was procured from the mines of Kongsberg, in Norway; this specimen, which is preserved in the Museum of Copenhagen, weighs about 500 lbs. Others of still larger size have been cited as coming from the same locality. Crystals of native silver are seldom very distinctly defined, as they do not usually occur in an isolated state, and are generally more or less distorted. The cube, the octahedron, and the dodecahedron, are among the forms which it most commonly assumes. Native silver is often found disseminated through ferruginous veinstones or "gozzans," as at Huelgoët, in Brittany, and in the mines of Chili, Peru, and Mexico, where such ores receive the names of "pacos" and "colorados." Native silver is also found in the Harz, Saxony, Hungary, and Dauphiny; large quantities are afforded by the mines of Peru and Mexico; and in the United States of America beautiful specimens are found associated with the copper procured from the district south of Lake Superior. Native silver is usually alloyed with some gold, copper, or iron, that of Kongberg contains about 5 per cent. of mercury.

**NATIVE AMALGAM**; *Argent amalgamé; Silberamalgam.* Cubic.—This mineral, which is of a silver-white colour and bright metallic lustre, occurs both in distinct crystals and amorphous; it also not unfrequently assumes the form of thin plates, occupying fissures in the veinstone.

Its specific gravity is about 14; when heated before the blowpipe, mercury is expelled, and a fused button of metallic silver remains. The composition of this amalgam is, according to Klaproth—

| | |
|---|---|
| Ag | 36·00 |
| Hg | 64·00 |
| | 100·00 |

This corresponds with the formula AgHg; another variety, with 74 per cent. of mercury, is $Ag_2Hg_3$. This mineral is found in many localities, but the finest specimens have been procured from Moschellandsberg in Rhenish Bavaria.

Richer amalgams, called *Arquerite*, form one of the sources of silver in the rich mines of Arqueros, in the province of Coquimbo, Chili. From their malleability and general appearance, they were long thought to be metallic silver. According to the analysis of Domeyko, of the Mining School, Coquimbo, they consist of—

| | | |
|---|---|---|
| Ag | 86·50 | 65·4 |
| Hg | 13·50 | 34·6 |
| | 100·00 | 100·0 |

which correspond to $Ag_{12}Hg$ and $Ag_7Hg_2$.

**ARGENTITE**; **SULPHIDE OF SILVER**; *Argent sulfuré; Silberglanz; Glaserz.* Cubic.—Occurs massive, and crystallized in cubes and dodecahedra. Its colour is a blackish lead-grey, with a shining streak of the same colour. The fracture of the massive varieties is slightly conchoidal. It is fusible even in the flame of an ordinary candle, and before the blowpipe, on charcoal, gives off sulphurous vapours, and yields a button of silver. This is at the same time one of the richest and most abundant ores of silver, and furnishes a large proportion of that annually produced by various foreign mines. It occurs in those of Saxony, Bohemia, and Hungary, and is particularly abundant in the mines of Guanaxuato and Zacatecas, Mexico. It occurs abundantly with stephanite, native gold, &c., in the Comstock lode, Nevada.

The composition of argentite is, silver 87 and sulphur 13 per cent., corresponding to $Ag_2S$.

**Stephanite**; **Brittle Silver Ore**; *Psaturose; Sprödglaserz.* Rhombic. —This mineral is of an iron-grey colour, inclining to black, with metallic lustre and unequal conchoidal fracture. It has a specific gravity of 6·2, is extremely brittle, and when broken yields a black powder.

Before the blowpipe, on charcoal, it affords a button of silver, after having given off sulphurous and antimonious fumes, in which the peculiar odour of arsenic may often be detected. When crystallized, it is usually found in small, flat, six-sided prisms.

This mineral, which occurs with other silver ores at Freiberg, Schneeberg, and Johanngeorgenstadt, as well as in the mines of Bohemia and Hungary, and in those of Chili, Peru, Mexico, and Nevada, has, according to analyses of Rose and Kerl, the following composition :—

| | From Andreasberg. By Kerl. | From Schemnitz. By H. Rose. |
|---|---|---|
| Ag . . . . . | 68·38 | 68·54 |
| Cu . . . . . . | ... | 0·64 |
| Fe . . . . . . | 0·14 | ... |
| Sb and As . . . | 15·79 | 14·68 |
| S . . . . . . | 16·51 | 16·42 |
| | 100·82 | 100·28 |

Formula $Ag_5SbS_4$, or $5Ag_2S.Sb_2S_3$.

**Polybasite**; *Polybasite; Eugenglanz.* Rhombic.—Crystals usually short tabular prisms, with triangularly striated base, parallel to alternate edges; cleavage basal, imperfect. Also massive and disseminated. Colour, iron-black, excepting when in thin crystals, which appear cherry-red by transmitted light; streak black. Specific gravity, 6·21.

Heated in an open tube it fuses, giving off sulphurous and antimonious fumes; on charcoal, before the blowpipe, fuses into a globule with evolution of sulphur, and sometimes arsenic, coating the support with an antimonious deposit. It occurs in the mines of Guanaxuato and Guadalupe-y-Calvo, in Mexico; in Chili; at Freiberg; and at Przibram, &c. Also in Nevada, Idaho, and various other localities in the United States.

Two specimens of this mineral, analysed by H. Rose, afforded the following results :—

| | From Mexico. | From Schemnitz. |
|---|---|---|
| Ag | 64·29 | 72·43 |
| Sb | 5·09 | 0·25 |
| As | 3·74 | 6·23 |
| Cu | 9·93 | 3·04 |
| Fe | 0·06 | 0·33 |
| Zn | ... | 0·59 |
| S | 17·04 | 16·83 |
| | 100·15 | 99·70 |

The second is approximately $8Ag_2S.Cu_2S.As_2S_3$.

**Dark-Red Ruby Silver Ore; Pyrargyrite**; *Argent rouge antimonial; Dunklesrothgültigerz.* Rhombohedral.—Colour black to cochineal-red; streak red; transparent to opaque; fracture conchoidal. Specific gravity 5·7–5·9.

**Light-Red Ruby Silver Ore; Proustite**; *Argent rouge arsenical; Lichtesrothgültigerz.* Rhombohedral and generally similar to Pyrargyrite, but of a lighter red colour and adamantine lustre.

These minerals are analogous in constitution, and of the following composition:—

| | Pyrargyrite, $Ag_3SbS_3$. | Proustite, $Ag_3AsS_3$. |
|---|---|---|
| Ag | 59·78 | 65·46 |
| Sb | 22·51 | ... |
| As | ... | 15·15 |
| S | 17·71 | 19·81 |
| | 100 | 100 |

Fused with sodium carbonate on charcoal, before the blow-pipe, either mineral affords a globule of silver.

Ruby silver ores occur with calcite, native arsenic, and galena at Andreasberg, in the Harz; also in Hungary, Saxony, Norway, at Guadalcanal in Spain, &c., and are abundant in Mexico. Beautiful crystals of Proustite have been obtained in Chili from mines near Copiapo, and it also occurs in the Austin district, State of Nevada; in the Poor Man lode, Idaho, it has been found in masses weighing several pounds.

*Freieslebenite* or *Schilfglaserz* ($Ag_3Pb_2Sb_3S_8$), containing 22·5 per cent. of silver, which occurs as a rarity in Saxony and Bohemia, has been obtained in large quantity at Hiendelaencina in Spain.

*Stromeyerite* and *Jalpaite* are isomorphous mixtures in various proportions of $Cu_2S$ and $Ag_2S$, containing from 3 to 70 per cent. of silver. The first is rhombic and the second cubic in crystalline form. They are of some importance as ores in Arizona and Northern Mexico.

*Tetrahedrite*, especially in the purely antimonial varieties, found principally in the Harz, the Isle of Man, Cornwall, Tyrol, Peru, Chili, and Bolivia, contains from 10 to 30 per cent. of silver. The mixed varieties, with both antimony and arsenic, contain less silver, and the pure arsenical kinds, none.

CHLORIDE OF SILVER; CERARGYRITE; *Argent corné; Silberhornerz.* Cubic.—This mineral constitutes one of the richest and most abundant ores of Chili and Peru, where it is frequently associated with native silver, apparently resulting from its decomposition. It also occurs in amorphous fragments in connection with sulphide of silver, but still more frequently disseminated in the ferruginous ores known in Perú under the name of *pacos*, and in Mexico as *colorados.* The chloride of silver of Huelgoët is of this description, and is disseminated through a cavernous hydrated iron oxide, and sometimes assumes the form of small octahedral crystals, the largest of which do not exceed in size the head of an ordinary pin. Very large masses have been obtained at Leadville, Colorado.

The colour of this mineral is white or yellowish-white, which becomes violet by exposure to light; massive fragments, when broken, present a conchoidal fracture, the edges being transparent or translucent. Chloride of silver is extremely soft, and admits of being cut with a knife or scratched by the nail. It is fusible before the flame of the blow-pipe, and, when supported on a piece of charcoal, affords a pearl-like button, which, by continued exposure to the reducing flame, yields a globule of silver. On being moistened with water, and rubbed with a piece of metallic zinc, its surface becomes covered with a film of reduced silver. Chloride of silver, when pure, consists of silver 75·3, chlorine 24·7; and its composition is therefore represented by the formula $AgCl$. Sp. gr. = 4·45.

*Embolite*, or chloro-bromide of silver, very commonly occurs in association with the preceding mineral in Mexico, Chili, and Colorado. It is usually in crystalline grains and small masses of a greenish colour. The composition is not well defined, as bromine and chlorine replace each other in every proportion. It is doubtful whether pure bromide of silver occurs in nature.

*Iodide of silver* ($AgI$) has been found in Chili and Mexico.

Several combinations of silver with selenium and tellurium, as well as alloys with bismuth and antimony, are found in nature, but only as mineralogical rarities; and the preceding list includes all the more important minerals containing silver as a principal constituent. These, also known as *precious* or *noble* silver ores, are only exceptionally found in large masses, the ores of even the richest silver mines consisting as a rule of intimate mixtures of such precious minerals with quartz, calcite, barytes, and other earthy minerals, iron- copper- and arsenical-pyrites, nickel and cobalt ores, zinc-blende, galena, and their oxidation products, which cannot be subjected to any dressing process without great loss, so that the classification of the ore is mainly done by hand picking. A small proportion of very rich mineral with 400 to 2,000 ounces or more of silver per ton may thus be obtained, but the bulk of workable ores contains very much less, and may probably be taken as ranging from about 20 up to 200 ounces per ton, the lower profitable limit varying, of course, with the facilities for treatment and other local circumstances.

*Dry ores*, or those consisting essentially of quartz and rich silver minerals, are also known as *milling* ores, *i.e.*, suitable for reduction in amalgamating mills, while those containing sulphides of lead or copper in quantity are *smelting* ores.

*Free milling* ores are dry ores without sulphur, arsenic, or antimony. When these substances are present they are *refractory* ores, which should be roasted before amalgamation.

Refractory smelting ores are those containing antimony, arsenic, and more particularly zinc-blende. The latter mineral is extremely prejudicial to the silver smelter, and special reductions are sometimes made on the price of ores when the proportion of zinc exceeds 15 to 20 per cent.

## DISTRIBUTION OF SILVER ORES.

The rocks enclosing silver ores differ very widely both in age and composition. Silver ores often occur in lodes enclosed in the older crystalline and metamorphic rocks, and many of these have been worked to great depths without permanent change in character or apparent diminution of productiveness. Some of the veins of Norway and Saxony are remarkable examples of this mode of occurrence. In South America a large proportion of the silver is obtained from veins in calcareous rocks. The silver-bearing strata of Peru and Bolivia are believed to be of Carboniferous age; whilst the rocks, in which the famous silver mines of Chañarcillo and Trespuntas, in Chili, are worked, are said to be Lower Cretaceous. Those enclosing the celebrated Comstock vein at Virginia City, in the State of Nevada, are eruptive, belonging, according to Richthofen, to the latter part of the Tertiary and beginning of the Post-Tertiary periods.

Nearly the whole of the silver obtained in the United Kingdom is extracted from argentiferous lead ores by processes which have been described when treating of the metallurgy of lead; the amount thus annually produced is now only about 28,000 lbs. troy.

The celebrated Kongsberg mines, in Norway, were discovered in 1623, and have been worked with but little interruption up to the present time; their yield of silver in 1879 was 10,500 lbs., but they were formerly much more productive. Sweden also produces a small amount of this metal from the mines of Sala, in Westmanland.

Austria-Hungary annually produces about 68,000 lbs. troy of silver, while Saxony yields 161,326 lbs., of which amount Freiberg alone affords 157,600 lbs. The annual yield of the Upper and Lower Harz is about 127,500 lbs. The production of silver at Mansfeld in 1884 was 198,303 lbs., and the total production of the German Empire during the same year was 660,524 lbs. troy.

The annual production of Russia is estimated at 27,000 lbs. troy. In France there are, at the present time, no mines in operation on what can be regarded as silver veins, although those of La Gardette and Chalanches, in the department of Isère, were formerly worked for silver ore. Huelgoët, in Brittany, although strictly speaking a mine of argentiferous galena, yielded an ochreous clay containing about 30 ounces of silver, in the form of chloride, per ton of ore. The mines of Pontgibaud, Puy-de-Dôme, produce argentiferous galena, very rich in silver, which is extracted by smelting in low blast-furnaces. The yearly production of silver in France is estimated at 16,000 lbs. The annual yield of Italy, chiefly from the ores of Sardinia, is about 70,000 lbs.

In Spain silver mines were formerly worked in mica-slate at Guadalcanal and Cazalla, north of Seville, and were, at one time, very productive. Antimonial lead ores rich in silver are found in the Sierra Almagrera, near Almeria. For some years the most important silver mines of Spain were those of Hiendelaencina, in the province of Guadalajara, seventy miles north of Madrid. They were discovered in 1843 by a native of the district, who had worked in the mines of Mexico. This man, on his return to his native village, remarked the resemblance which a large stone on the roadside bore to some of the silver ores he had seen in the mines of the New World. On being analysed it was found to be a rich ore of silver, and the stone proved to be the out-crop of a regular vein. The ore at Hiendelaencina is chiefly freieslebenite. The production of these mines from 1846, when they were first worked, up to June 1866, amounted to 631,544 lbs. troy, but their returns since the year 1858 have been very small. The present annual silver production of Spain, including that extracted from pyrites, but which is mostly obtained from lead, is estimated at about 250,000 lbs. troy.

Formerly the most important mines of North America were those of Mexico. But of late years enormous amounts of silver bullion have been produced in the United States.

At the date of the publication of Humboldt's 'Essai Politique' (1825), the mining districts of Mexico, arranged in accordance with the importance of their several yields, followed each other in the subjoined order:—

| | | |
|---|---|---|
| Guanaxuato, | Intendency of | Guanaxuato |
| Catorce | ,, | San Luis Potosi |
| Zacatecas | ,, | Zacatecas |
| Real del Monte | ,, | Mexico |
| Bolaños | ,, | Guadalajara |
| Guarisamey | ,, | Durango |
| Sombrerete | ,, | Zacatecas |
| Tasco | ,, | Mexico |
| Batopilas | ,, | Durango |
| Zimapan | ,, | Mexico |
| Fresnillo | ,, | Zacatecas |
| Ramos | ,, | San Luis Potosi |
| Parral | ,, | Durango |

It is stated, by the same author, that the silver extracted from the mines of Mexico, from 1st January 1785 to 31st December 1789, amounted to 7,314,344 lbs. troy. The War of Independence caused a great falling-off in the annual production of the precious metal, which between the years 1810 and 1845 probably did not average more than 800,000 lbs. of silver and a little over 400 lbs. of gold. Since 1850, however, the mines of Mexico have partially regained their ancient prosperity, and their present annual produce of silver may be estimated at 1,000,000 lbs. This is mainly derived from localities in the central provinces, as given in the above list, but the northern mines in Sonora and Chihuahua are rising in importance.

Although the discovery of silver in the State of Nevada only dates from the year 1859, its extraordinary production for more than twenty years rendered its mines more famous than those of countries in which silver mining has been carried on for ages. The mines on the great Comstock vein in the vicinity of Virginia City have been very extensively worked, and up to the end of June 1880 produced bullion to the value of £63,000,000, of which value £28,000,000 was represented by gold. In addition to those on the Comstock vein, which now appears to be nearly exhausted, there are silver mines in the Austin district, at White Pine, and in other portions of the State. The ores found on the Comstock chiefly consist of argentite and stephanite, whilst many of the mines in the neighbourhood of Austin yield dark-red silver ore. At White Pine the ores were chiefly chlorides. Idaho has afforded massive pyrargyrite. The other most important silver-producing States are Colorado, Utah, Montana, Arizona, California, and New Mexico.

Native silver is also found associated with the native copper of Lake Superior. The annual production of silver in the United States and Territories is now not less than 2,650,000 lbs. troy.

A small mine, producing very rich ores like those of Kongsberg, was worked for several years at Silver Islet, Lake Superior, and sulphide of silver has latterly been found in some quantity at Rabbit Mountain, also on the Canadian side of the lake.

Comparatively little is known of the geology of Central America, excepting that the predominant rocks are granite, gneiss, and mica-slate, and that the abundance of igneous rocks bears witness to extensive volcanic action. Silver is found in various parts of that country, and about twenty-five years since some mines, worked by an English company, in the mountain of Alotepec, afforded, in the course of eight years, 53,000 lbs. of silver.

The silver mines of Peru, Bolivia, and Chili are of considerable importance. At the mines of the Cerro de Pasco, in Peru, the principal ores worked are of the description known as "pacos," which are analogous to the *Gossans* of the Cornish miner. These argentiferous gossans have been found in such enormous quantities at Pasco, where they have been worked since the commencement of the seventeenth century, that although during the last twenty years of the eighteenth century they had produced above five million marks of silver, but few of the workings had penetrated to a depth of a hundred feet. In addition to the Cerro de Pasco there are various other districts in Peru which have produced large quantities of silver. The total annual production of the silver mines in Peru is about 230,000 lbs. troy.

The mining district of Potosi, which was once included in the vice-royalty of Buenos Ayres, now forms a portion of Bolivia. In this locality above thirty principal veins, besides others of less importance, have been worked, and have at various times, since their discovery in 1545, yielded almost fabulous amounts of silver. Many of them are situated in a mountain called *Hatun-potocsi*, the summit of which reaches an elevation of 16,000 feet above sea-level; the ores obtained, from the period of their discovery up to 1571, when the process of amalgamation was first introduced, were treated exclusively by fusion. The period of the greatest productiveness of the Potosi mines was during the century immediately following their discovery, when the average annual amount of the silver produced was about 600,000 lbs. troy. Shortly after the commencement of the seventeenth century their yield began to decline, and at its close, only amounted to an annual value of from six to eight hundred thousand pounds. Rich deposits of silver ore have also been discovered in the district of Caracoles, now ceded to Chili. The quantity of silver ore exported from Antofagasta and Mejillones de Bolivia (the ports of Caracoles) from November 1872 to July 1873 was 10,115 tons, containing about 300,000 lbs. of silver. The most important Bolivian mine is at Huanchacha, about 80 miles S.W. of Potosi on the Atlantic slope of the Andes. The present annual yield of Bolivia is estimated at from 448,000 to 460,000 lbs. troy.

The most important silver mines of Chili are those situated in the neighbourhood of Copiapo. The principal workings are in the vicinities of Chañarcillo and Trespuntas, the first sixteen leagues south, and the second thirty to the north-east, of Copiapo. In Chili the development of the mineral resources of the country has been of more recent date than in the other South American States; but its comparatively flourishing political situation has enabled workings to be established on an extensive scale, and within a few years a considerable increase has taken place in the amount of silver produced. Its present yield is estimated at about 400,000 lbs.

The Santa Anna mines in the province of Mariquita, New Granada, were formerly of some importance, producing from 11,000 to 12,000 lbs. of silver annually.

It is estimated that the total annual production of the known silver-producing countries of the world is now about 7,650,000 lbs. troy.

## ASSAY OF SILVER ORES.

BY FUSION WITH LITHARGE, &C.—In assaying silver ores, the object sought is to obtain the silver in the form of an alloy with lead, which is afterwards passed to the muffle and cupelled in the manner described when treating of the method of assaying lead ores containing silver.

Argentiferous minerals containing copper may be assayed either by scorification, or by fusion with oxide of lead, as if silver alone were present, since the resulting button of alloy admits of being cupelled with a proper addition of metallic lead. The roasting of such ores, when necessary, should be effected at as low a heat as possible, as, from their great fusibility, they would otherwise be liable to agglutinate, by which the complete expulsion of sulphur would be rendered difficult.

Ores of silver in which the metals (after roasting or otherwise) exist in the form of oxides are commonly fused with a mixture of litharge, or red lead, and finely powdered charcoal, by which an alloy with lead is obtained, which is subsequently subjected to cupellation. The proportion of litharge employed for this purpose must be varied according to circumstances, as the resulting button of alloy should neither be too rich, since in that case a portion of the silver might be lost in the slag, nor too poor, as the cupellation would then occupy a long time, and undue loss of silver through sublimation and absorption by the cupel would be entailed. In ordinary cases, if 400 grains of ore be the quantity operated on, a button of 200 grains will be a convenient amount for cupellation; and this may be obtained by the addition of 500 grains of litharge, and from 12 to 15 grains of finely powdered charcoal or lamp-black. The whole is to be well mixed with 200 grains of sodium carbonate, on a sheet of glazed paper, and afterwards introduced into an earthen crucible, of which it should not fill more than three-fifths the capacity. This is covered with a thin layer of borax, and fused in an ordinary assay-furnace, care being taken to withdraw it from the fire as soon as a liquid and perfectly homogenous slag has been obtained, since the unreduced litharge would otherwise be liable to cut through the pot and thus spoil the results. When it has sufficiently cooled, the crucible is broken, and the button of alloy obtained passed to the cupel.[1] In this and all similar cases it is necessary to ascertain by previous experiment the amount of silver contained in the lead resulting from the reduction of a given weight of the litharge employed, in order to be possessed of data from which to make the requisite deduction from the result obtained. With very poor litharge, however, the resulting lead contains so small an amount of silver, that, for many commercial purposes, its presence may be neglected.

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[1] Instead of breaking the crucible its contents may be poured into a cast-iron mould.

When other minerals than oxides or carbonates are to be examined, the addition of charcoal, or any similar reducing agent, becomes in some instances unnecessary, as litharge readily attacks the sulphides, &c., and oxidizes nearly all their constituents, with the exception of silver, whilst a proportionate quantity of metallic lead is reduced. The slags formed in this way contain the excess of litharge added, and the button of alloy produced is subjected to cupellation in the usual manner. The proportion of oxide of lead added to ores of this description must vary in accordance with the amount of oxidizable substances present. It should, however, in all cases be in excess, since, if the slags retain any undecomposed sulphide, the whole of the silver contained in the ore may not be collected in the button of alloy obtained.

An objection to this method of assaying is to be found in the large amounts of lead produced for cupellation; as one part of pure iron pyrites reduces 8½ times, and sulphide of antimony or grey copper ore from 6 to 7 times its weight of lead. This inconvenience may be obviated by effecting the partial oxidation of the mineral either by roasting or by the aid of nitre, by the skilful use of which the button of lead may be obtained of almost any required weight.

When this reagent is employed in excess, it determines the oxidation of all the oxidizable constituents of the ore. But when, on the contrary, the mixture contains a large amount of litharge, and nitre has not been added in sufficient quantity to decompose the whole of the sulphides present, reaction takes place between the undecomposed sulphides and oxide of lead. This gives rise to the liberation of metallic lead, which, combining with the silver, affords a button of alloy, from which the latter metal is obtained by cupellation. The amount of nitre employed for this purpose will depend on the composition of the ores operated on, but it should be borne in mind that 2½ parts of potassium nitrate are sufficient to completely oxidize 1 part of pure iron pyrites, and that 1½ and ⅔ their respective weights are, in the case of sulphide of antimony and galena, sufficient to produce the same effect.

Scorification.—This is a simple and convenient process for assaying silver ores, as well as some gold ores.

It consists in exposing the finely ground ore, mixed with granulated lead, and placed in a saucer-shaped scorifier, to the action of a bright red heat, in an ordinary assay muffle.

Part of the lead is thus converted into litharge, and this, as fast as it is produced, reacts on the various substances contained in the ore, forming with them a clean slag, in which no appreciable amount of the precious metals is met with. Practically, the whole is found alloyed with the lead remaining after the operation. The saucer-shaped vessels or scorifiers, employed for this purpose, should be of close-grained fire-clay well baked. It is important that they should be compact in structure, so as to resist the corrosive action of fused litharge. They should also be capable of bearing sudden changes of temperature without cracking.

A sufficient number of these scorifiers having been selected for the assays to be made, 100 grains of the ore, ground to a fine powder, and carefully dried to expel moisture, are taken and intimately mixed with a certain quantity of granulated lead, and a small proportion of pounded borax—both being previously placed in the scorifiers, arranged in order on the assay table. The proportion of lead added varies in accordance with the greater or less refractory nature of the ore; namely, from five to eight times the weight of the material operated on. In all cases, however, it is advisable to add an excess of lead, as the slags are thereby rendered more liquid.

The lead used should, if possible, be free from silver, but, in many cases, such lead cannot be obtained. It is consequently requisite to estimate beforehand the amount of silver present and make a corresponding deduction from the weights of the buttons afforded by assay.

The scorifiers being charged with a due proportion of ore, lead, and flux, and the muffle brought to a full red heat, they are removed to the furnace, and as many introduced as there may be room for in the muffle. The introduction of the scorifiers at first considerably reduces the temperature of the muffle, and, in consequence, pieces of charcoal may, with advantage, be placed in its mouth to assist in again raising the heat. The door of the muffle is now closed, and in a few minutes the lead enters into fusion. White vapours are seen rising from the assay, and the formation of litharge rapidly takes place. As the borax melts, and the quantity of litharge increases, the whole mass in the scorifier softens. With a further increase of temperature, it becomes liquid, and the lead collects in a large globule in the centre. When the assay has become thoroughly heated, which generally occurs in ten or fifteen minutes from the commencement of the operation, the door of the muffle is removed, air enters in greater quantity, and oxidation of the lead proceeds more rapidly.

As litharge accumulates, the slag, formed by its combination with siliceous and other matters contained in the ore, is increased in quantity, and gradually extends itself over the surface of the lead. The mouth of the muffle is now allowed to remain open for ten or fifteen minutes. It is then closed, and the temperature raised to bright redness for about five minutes, in order to render the slags as liquid as possible before pouring, and to facilitate the union of any disseminated globules of alloy.

The scorifiers are now withdrawn from the muffle by proper tongs, and their contents rapidly poured into suitable moulds. When cold, the buttons of lead are separated from the adhering slags by a few blows with a hammer. The lead obtained should be soft and ductile. If it be at all brittle, either an insufficient quantity has been used, or the scorification has not been carried sufficiently far to allow of complete fluxing of the impurities. When the operation has been successfully conducted, the buttons contain practically the whole of the precious metals present in the ore, and must be subsequently treated by cupellation.

It is essential that the slags should be uniformly liquid at the time of pouring from the scorifier. If they be hard or contain pasty lumps, a portion of the mineral may be left unacted upon, and small metallic buttons may be enclosed in the lumps, or remain attached to the sides of the scorifier. If the slags should not appear perfectly liquid, when a sufficiently high temperature is maintained in the muffle, and the other conditions of the process have been attended to, it will be necessary to add more borax. In some cases it may be requisite to stir the slags with an iron rod, in order to divide the lumps which have been formed, and to incorporate them with the more liquid slags.

This method of assay is applicable to all kinds of argentiferous and auriferous ores, without exception, when of moderate richness, and from its convenience and the short time required, it is frequently employed in establishments where a great number of assays have to be made daily. When, however, poor ores have to be examined, fusion in a crucible with litharge is preferable, since, by that method, a greater quantity of mineral can be operated on, and consequently more accurate results obtained.

## ASSAY OF SILVER BULLION.

FIRE ASSAY.—The assay of silver bullion, by the dry way, is conducted as follows:—First, a fair sample of the alloy to be operated on is accurately weighed in a delicate balance; secondly, this weighed portion of the bar is cupelled with lead; and, thirdly, the button of silver remaining on the cupel is weighed. The difference between the two weighings will represent the amount of impurity which has been removed. The furnace employed for this purpose may either be of fire-clay bound with hoop-iron, or may be constructed of a wrought-iron shell lined with fire-tiles, as shown in figs. 175, 176; in either case, it must be provided with an arrangement for easily regulating the draught.

In order to save trouble in computing results, special weights are generally used in bullion assaying. The largest, being the quantity usually taken for an assay, is called the assay pound; while the smaller ones represent troy ounces, pennyweights, and half-pennyweights, the latter being the lowest denomination to which silver bullion is reported. Thus, with an assay pound of 12 grains, the ounce, dwt., and half dwt., are represented by 1 grain, 0·05 grain, and 0·025 grain respectively. When metrical weights are used with assay pieces of one gramme,[1] the weight of the result in milligrammes gives the millesimal fineness, or the proportion of silver, in one thousand parts of the alloy, without further calculation.

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[1] When the alloy to be examined is over 800 fine, 1 gramme is generally employed; if under 800, half a gramme only is taken.

In England the basis of silver bullion valuation is sterling or standard silver, as used for coinage, containing 11 ounces 2 dwts. of silver and 18 dwts. of alloy in the troy pound, and assays are reported as better or worse than standard in dwts. per pound according to the excess or deficiency of alloy. The corresponding millesimal fineness is 925 silver and 75 alloy, and therefore $75 \div 18 = 4\frac{1}{6}$ per 1000, is equivalent to 1 dwt. per lb. For instance, French or American coin of 900 silver and 100 alloy, is $100 - 75 = 25$ per 1000, or 6 dwts. "worse," and bars of 932 fine are $75 - 68 = 7$ per 1000, or 1½ dwts. "better" than standard silver.

In sampling an assay of silver bullion the first step to be taken is to flatten out the cuttings taken from the several bars. The edges of these are removed by the use of a pair of shears, and from each are prepared two representative assay pounds, which are carefully put aside with one-half the lead necessary for effecting their cupellation.

The amount of lead necessary to be added to an alloy of silver and copper varies in accordance with the composition of the mixture to be treated, and should be greater in proportion as the quantity of copper becomes more considerable. In making this addition, it is necessary to bear in mind that the lead must be present in such quantity that the litharge formed may be enabled to dissolve the other oxides produced, and at the same time remain sufficiently liquid to be absorbed by the cupel. If the amount added be insufficient, the litharge formed becomes pasty, and speedily covers the surface, while, if too large a quantity be employed, the assays remain a long time in the muffle, and an undue loss of silver by volatilization is experienced.

English standard silver requires about six times its weight of lead for cupellation. The affinity exercised by silver for copper renders it necessary, in these operations, to add a larger amount of lead than would be required if pure copper were to be dealt with.

The following table shows the amounts of lead necessary to effect the cupellation of various mixtures of silver and copper:—

| Amount of Silver. | Amount of Copper. | Quantity of Lead required for One Part of the Mixture. |
|---|---|---|
| 1000 | 0 | 1 |
| 950 | 50 | 3 |
| 900 | 100 | 7 |
| 800 | 200 | 10 |
| 700 | 300 | 12 |
| 600 | 400 | 14 |
| 500 | 500 | 16–17 |
| 400 | 600 | |
| 300 | 700 | |
| 200 | 800 | |
| 100 | 900 | |
| Pure copper | 1000 | |

The lead employed should be as free as possible from silver, and the amount of that metal thus introduced into the assay must be deducted from the results obtained. The duplicate weighed portions of the several bars of bullion to be assayed must be wrapped in lead-foil, and arranged in the compartments of a divided tray, having nearly the dimensions of the bottom of the muffle in which the cupellations are made, so that the place of each may correspond with its position in the furnace.

When the muffle has attained a uniform bright red heat, the cupels are introduced, and its mouth is closed for a short time, either by a door of fire-clay or by a large piece of charcoal, when the temperature of the furnace has been considerably lowered by the introduction of cold cupels. As soon as the latter are raised to the proper bright-red heat, one-half of the lead necessary for cupellation is introduced by the aid of suitable tongs; and, when it has become fused, the assays, wrapped in the other half of the lead required, are dropped into each cupel in the order in which they stand in the divided tray. When all have been arranged in their proper places, the assayer closes the mouth of the muffle for a short time, and shortly afterwards opens it to a greater or less degree, for the purpose of admitting a current of air, by which the starting of the cupellations is effected. When this takes place the metallic bath becomes uncovered, with the exception of some small patches of fused litharge, which move rapidly from the centre towards the edges of the cupel. As soon as the working is thus fairly begun, the draught must be adjusted by more or less completely closing the various openings in the furnace so as to maintain a uniform temperature in all parts of the muffle. When all is working satisfactorily but little fume should arise from the assays, and the litharge formed must be absorbed by the cupel as fast as it is produced. In this way the metallic globule goes on steadily diminishing in size until, after the expiration of about twenty minutes, the lead and baser metals have become oxidized and absorbed by the cupel, and the brightening of the silver takes place.

If the operation has been properly conducted, the "going-off" of the assays will commence with the first row, and successively pass back from row to row to the last. If, on the contrary, the working-off takes place irregularly, or commences at the far end of the muffle, it is an indication that the draught has not been skilfully regulated; and in such cases the results obtained are usually less to be depended on than if the brightening of the buttons had taken place regularly from the mouth of the muffle towards its further extremity. The several assays, as soon as they have gone off, are drawn to the mouth, and, after being allowed to cool slowly, are removed into a divided iron tray, in the order in which they were placed during the operation.

When the buttons thus obtained are round on the top, or are only slightly depressed in the centre, and can be easily removed from the cupels, it is an evidence of the operation having been properly conducted. If, on the contrary, they adhere firmly to the cupels, or throw out projections near the bottom, they are not perfectly fine. When they exhibit a flattened appearance, it is an indication that they have been cupelled with an insufficient amount of lead.

The buttons, when cleaned in the manner described on page 586, are weighed in order to determine the diminution of weight on that of the assay pieces. The difference of weight does not, however, give the proportion of alloy directly, as there are always small losses of silver due to volatilization and absorption by the cupel. These are determined by placing in the muffle, with each batch of assays, one or more *proofs* or check assays of bullion of known composition, which invariably show some loss when their known fineness is compared with that apparent after cupellation. A quantity equal to this loss is added as a correction to the assays made at the same time; as it is assumed that they will be all similarly affected.

Other metals, besides lead and silver, when present, usually afford indications on the cupel from which it is easy to judge of their nature, and, roughly, also of the amount in which they exist in the bullion. Pure lead stains the cupel of a straw-yellow colour, sometimes verging on orange. Copper gives a grey or dark-brown tint, according to its quantity. Iron produces a black stain, which is chiefly formed shortly after the commencement of the operation, and gives rise to a dark ring around the sides of the cupel. Zinc leaves a yellowish stain, and produces, during the process of cupellation, a luminous white flame and abundant fumes of the same colour, which carry off with them a notable amount of silver. Tin produces a grey slag, and antimony leaves spongy yellow scoriæ, which cause the circumference of the cupel to effloresce and to split off. The two last-named metals render the cupellation of the mixtures in which they exist difficult, and necessitate the employment of a large quantity of lead in order to carry off the infusible oxides formed. When assays of alloys containing silver have to be made by cupellation, it will be found convenient to keep in the laboratory a supply of lead-foil, ready weighed out into pieces of 1 and 2 grammes, whilst poor sheet-lead is kept in pieces of from 1 up to 6 grammes. Instead of weighing the larger pieces, they may be prepared with sufficient accuracy by casting in small bullet-moulds. By this means, the amount of silver in the lead being previously known, it becomes easy, by merely counting the number of pieces added, to know exactly what deduction is to be made at the termination of the cupellation for the silver in the lead.

VOLUMETRIC ASSAY.—On account of the difficulty experienced in obtaining perfectly accurate results by cupellation, a Commission was appointed by the French Government in 1829 for the purpose of examining the different processes then employed in the Parisian Mint for the assay of alloys containing gold and silver, and to report on any modifications it might be thought advantageous to introduce.

Gay-Lussac, who was one of the Commissioners to whom this question was submitted, proposed the adoption of the humid method of assay now generally employed, and published, in the name of the Commission, the details of the various necessary operations.

This process consists in determining the standard of the alloy examined by means of a solution of chloride of sodium, of which the strength has beforehand been accurately determined.

The solution of salt employed is so regulated that a decilitre is capable of exactly precipitating 1 gramme of pure silver. To determine the composition of an alloy, 1 gramme may be dissolved in 5 or 6 grammes of nitric acid, and to this is carefully added the standard solution of common salt from an accurately graduated pipette until the introduction of a fresh quantity ceases to be accompanied by a deposit of silver chloride. Towards the end of the experiment, when the point of saturation has been nearly arrived at, care must be taken to well shake the bottle after the addition of each successive drop of the saline solution, as by this means the liquor is rendered clear through the precipitation of the chlorine formed. When the whole of the silver has been thus thrown down, the number of divisions of the burette which have been employed in its precipitation are read off, and from the amount of chloride of sodium used the percentage of silver present is calculated.

When an accurate assay has to be made of an alloy of which the composition is beforehand approximately known, as in the case of silver coin or silver plate, the process is considerably simplified, and at the same time affords results of the most exact description. For this purpose two distinct solutions of common salt are employed; the first, or *standard solution*, is of such a strength that one decilitre will precipitate 1 gramme of pure silver; the second, called the *decimal solution*, is ten times weaker than the first, and consequently contains in a litre of solution the amount of chloride necessary to effect the precipitation of 1 gramme of pure silver.

The better to understand this process, let us suppose that an alloy intended for the French coinage is to be examined, which, in order to be accepted, should contain 0·897 of silver. We will assume that the alloy in question contains only 0·896 of silver, and consequently that 1·116 gramme of the mixture will correspond to 1 gramme of pure silver.[1] This quantity of the alloy is cut off, and, after being accurately weighed, is placed in a bottle, capable of being closed by a glass stopper, where it is dissolved in from 5 to 6 grammes of pure nitric acid; and, as soon as the solution has been effected, exactly 1 decilitre of the normal solution of salt is introduced.

It is evident that if, as was first supposed, the alloy really contained 0·896 of silver, the whole will be precipitated by the quantity of solution added, and that the supernatant liquor will contain no trace of chloride of sodium in excess. If, on the contrary, the proportion of silver is greater than that assumed, there will still remain a portion of that metal in solution; whilst, if it be less, the whole will have been completely precipitated, but the liquor will contain an excess of sodium chloride. To ascertain which of these results has been produced, the bottle is now carefully closed with its glass stopper, and briskly shaken, until the precipitated chloride has subsided and the solution has become clear.

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[1] When the amount of silver present is not known, a preliminary assay is first made by cupellation, in order to ascertain the approximate fineness.

When this point has been attained, one c.c. of the decimal solution of common salt, capable of precipitating 0·001 gramme of silver, is introduced. If any silver remain in solution, the liquor now becomes cloudy, and after being again shaken, another c.c. of the decimal solution is added. If, on the addition of this second c.c. of the solution, the liquor again becomes turbid, it is, after being well shaken, allowed to clear, and a third c.c. of the decimal solution poured in, and so on, until no further turbidity is produced on the addition of a fresh quantity of the decimal solution. If we suppose that five c.c. of the decimal solution, successively added, have produced a precipitate in the liquor, whilst the addition of the sixth has in no way affected its transparency, we may conclude that after the precipitation of 1 gramme of pure silver by the decilitre of the standard solution, the liquor still contained at least 0·004 gramme of silver. From the circumstance of the fifth c.c. of decimal solution having caused a turbidity, while the sixth produced no kind of effect on the solution, it is also evident that the liquor at most did not contain more than 0·005 gramme of silver; and therefore, in taking it at 4½ thousandths we arrive at the result to within 0·0005 of the truth. The proportion of silver in the alloy examined will therefore be $\frac{1{,}000 + 4\frac{1}{2}}{1{\cdot}116} = 900$ thousandths. When, on the contrary, a c.c. of the decimal solution gives no further precipitate in the solution of silver which has already received the decilitre of the standard liquid, it is evident that the silver in the alloy must be inferior to 0·896, and consequently the mixture is below the legal standard. If in this case its exact composition be required, recourse must be had to a standard solution of nitrate of silver, called the *decimal solution of silver*, so adjusted that one litre of the liquor may contain exactly 1 gramme of silver.

To use this, a c.c. of the decimal silver solution is first dropped from a pipette into the bottle containing the assay, and removes the chlorine contained in the same volume of the decimal solution of common salt, which was added for the purpose of ascertaining whether the whole of the silver had been precipitated. The liquor is now brightened by agitation, and another c.c. of the silver solution added. If a turbidity is produced the bottle is again shaken, and a third c.c. of the solution is introduced after the chloride formed has been completely deposited. This is continued until an addition of the silver solution ceases to cause a precipitate in the solution to be assayed. If, in this case, the first five c.c. of the silver solution (without counting that used to neutralize the effect of the c.c. of decimal solution of common salt, first added to ascertain if any silver remained in solution after the addition of the decilitre of standard solution, gave rise to the formation of a precipitate, and on the introduction of the sixth the liquor remained perfectly clear, it is probable that the fifth c.c. was not entirely decomposed. It is therefore customary to admit that 4½ c.c. of the silver solution have been sufficient to effect the decomposition of the excess of chloride of sodium remaining in the liquid after the introduction of the decilitre of standard solution.

It is consequently evident in this case, that it will be necessary to subtract 4½ thousandths, and that the correct composition will be expressed by $\frac{1{,}000 - 4\frac{1}{2}}{1\cdot116} = 892$ thousandths.[1]

In establishments where large numbers of assays of silver alloys are made daily, the apparatus is so arranged as to materially facilitate the performance of the various operations above described. In the French Mint, where this method of assaying was first employed, the apparatus shown in figs. 201 and 202 has been adopted.

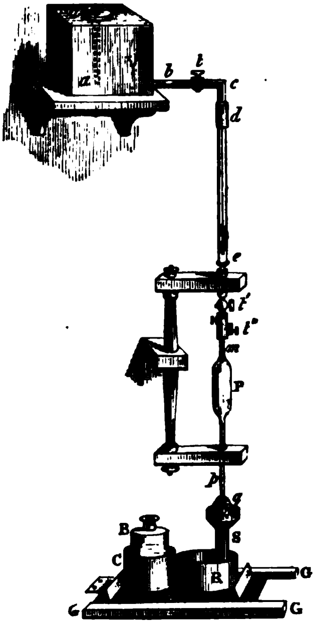

Fig. 201.

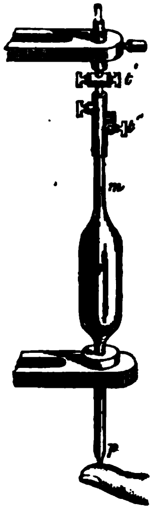

Fig. 202.

The standard solution of common salt is kept in a large vessel, V, made either of stoneware, or of sheet copper tinned on the inside. This reservoir, for the purpose of preventing evaporation, is covered by an immovable lid, provided with a tube, *a*, by which the air enters the chamber to supply the place of any portion of the solution that may be drawn off. This vessel, which is supported on a shelf fixed near the roof of the laboratory, is provided with a tube, *b*, *c*, *d*, bent at right angles at *c*, which admits of being closed by a silver stop-cock, *t*. The pipette, P, which contains exactly a decilitre of the liquid, is connected with the tube, *c*, *d*, by the glass tube, *d*, *e*, which contains a thermometer accurately graduated. The silver connecter by which the tube, *d*, *e*, is fastened to the pipette, P, is provided with two stop-cocks, *t′* and *t″*, of which the uses will be presently explained. In conducting an assay, the operator closes the extremity of the pipette with the forefinger of the left hand, fig. 202, and with the right opens the taps, *t′* and *t″*, the latter being opened first; by the first of these the solution enters the pipette, whilst from the second the air escapes in proportion as it becomes filled by the solution of sodium chloride. When the pipette has become filled by the liquor to a little beyond the mark, *m*, the cocks *t′* and *t″* are closed, and the instrument remains charged with the solution.

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[1] As chloride of silver is, to a certain extent, soluble even in weak solutions of sodium chloride, it is, in practice, usual, when the whole of the silver has been precipitated by the decilitre of normal solution, to add a given number of cubic centimetres of the decimal silver solution, and subsequently to estimate the amount of silver present, in solution, by the use of the decimal solution of common salt.

On the table immediately below this apparatus is a sliding support, W, in which is secured by a ring of copper, C, the bottle, B, containing the solution, in nitric acid, of the alloy to be assayed, while in connection with it is a small stand, S, on which is fastened a sponge, *q*, situated at the exact height of the beak, *p*, of the pipette. The assayer now slides the plate, W, between the guides, G, in such a way that the sponge may come in contact with the extremity of the pipette, and by carefully admitting air through an aperture in the tap, *t″*, allows the liquid to descend until it exactly reaches the level of the line, *m*, marked on the glass by a scratch-diamond. The sponge removes the last drop of the solution, which would otherwise remain attached to the beak of the instrument, and in proportion as it becomes saturated with moisture, the liquid falls through the hollow support, into the cup-shaped receiver, R. The operator now draws the slide towards the right until it is stopped by a peg which arrests it when the neck of the bottle is immediately under the extremity of the pipette, and by admitting air through the cock, *t″*, he allows the solution to flow into it. The last drop of the liquor invariably remains attached to the burette; but, as the instrument is gauged with due attention to this circumstance, its addition is unnecessary, and would vitiate the result. As in most instances several assays are being made at the same time, the weighed quantities of alloy are commonly dissolved in numbered bottles, which are arranged in a metallic frame somewhat similar to a cruet-stand, and which, after the introduction of the acid, may be placed in a water-bath or on a sand-bath for the purpose of facilitating their solution.

When the various samples have become completely dissolved, the nitrous fumes are removed from the bottles by slightly blowing into each, with bellows, through a glass tube, and a decilitre of the standard solution is introduced into each by the method already described. The bottles are subsequently placed in a metallic case, C, fig. 203, made with a cover to keep the stoppers in their places, which, besides being provided with compartments for each bottle, is suspended from the extremity of a steel spring, *a*, *b*, and is steadied from below by the elastic spiral, *c*, *d*. These bottles, after being carefully closed by their stoppers and fastened in their several compartments, are well shaken by an assistant, who takes hold of the handle, *e*, *f*, and briskly agitates the apparatus during several minutes. As soon as the liquors have in this way been rendered sufficiently clear, the bottles are removed from the frame, C, to a blackened table, fitted up with divisions numbered to correspond with the numbers on the bottles, care being taken that each assay be placed in the compartment to which it belongs. The decimal solution, which is contained in a phial having a pipette passing through its cork, is now employed for the purpose of determining the exact amount of silver in the various assays. This pipette is so marked, by a line on its surface, as to allow the operator to exactly measure out one c.c. of the liquid it contains. To do this the point of the forefinger is applied to the upper extremity of the tube, which, whilst thus closed, is raised above the surface of the liquid in the bottle, and is allowed to drop, by the careful admission of air, until the liquid has fallen to the level of the line marked on its surface. The opening is now closely stopped, and the c.c. of fluid is transferred to the first bottle of the series, into which it is permitted to flow on removing the finger from the upper extremity of the pipette.

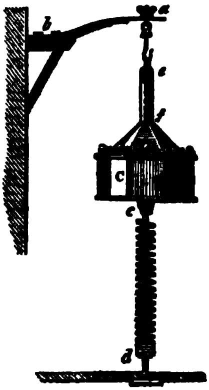

Fig. 203.

The same quantity of solution is afterwards successively added to each of the other assays. The assayer now examines each bottle in succession, and makes a mark with chalk on the blackened table before those in which a precipitate has taken place. These are a second time transferred to the shaking apparatus, in which they are briskly agitated until the liquids have again become clear, when they are taken back to their respective places on the blackened table, and another c.c. of decimal solution is added to each in which a precipitate was obtained by the last operation. By degrees the several bottles in which no precipitate has taken place are thus eliminated, and on counting the number of marks set before them the number of c.c. of the decimal solution which has been added to each assay is readily ascertained. From this number must be deducted half a c.c., as only a portion of the last addition may be supposed to have suffered decomposition.

The standard solution of sodium chloride employed is prepared at 15° C.; but as this, in common with all liquids, expands and contracts in accordance with the temperature to which it is exposed, it was formerly customary to construct a table of corrections, to be employed in cases when the liquid is used at temperatures either above or below this point. For this purpose the thermometers in the tube, *d, e*, was consulted, and the correction read off from tables prepared for that purpose. It is now generally preferred to make use of the following method, by which all error arising from any alteration in the solution is guarded against. With this object the assayer makes experiments daily on 1 gramme of pure silver, at the same time that he is conducting his regular assays of the usual alloys, and from the result obtained by these checks he is enabled to correct for any small irregularity in the temperature or constitution of the solution employed.

The standard solution of sodium chloride is made from common salt, without any preliminary purification, and is usually prepared in considerable quantities at a time. For this purpose 1 kilogramme of common salt may be dissolved in eight litres of water. The liquid is filtered, and the amount of water that would be necessary to make a solution of the requisite strength, supposing the chloride to be pure, is added. By this means a solution roughly approximating only to the composition of the normal liquor is obtained, of which the exact standard must be ascertained by adding a decilitre to a solution of 1 gramme of pure silver in nitric acid. The liquid is clarified by agitation, and by the addition of successive c.c. of the decimal solution, either of nitrate of silver or of sodium chloride, the exact amount of silver or of sodium chloride, as the case may be, remaining free after the addition of a decilitre of the solution, is ascertained.

When this is known it becomes easy to calculate the quantity of water or of sodium chloride which must be added in order to arrive at a correct standard; and when this addition has been made, experiments of a similar description are repeated, until satisfactory results are obtained. The decimal solution of sodium chloride is readily prepared by pouring a decilitre of the standard solution into a bottle of the exact capacity of a litre, and afterwards filling it with distilled water.

To prepare the decimal solution of silver, 1 gramme of pure silver is dissolved in nitric acid, to which distilled water is afterwards added until an exact litre of the liquid is obtained.

When the alloy operated on contains mercury, the results by humid assay are no longer exact, as this metal, being precipitated at the same time as the silver, decomposes a portion of the standard solution, by which the experiment becomes vitiated. The presence of mercury in the alloy examined is detected by exposing the bottles containing the precipitated silver chloride to the action of light, since the presence of a very minute trace of mercury prevents the usual darkening of that salt. The assay of alloys containing mercury may, however, be made by the humid process, if a solution of acetate of sodium be added to the solution of silver previously to the introduction of the standard solution of sodium chloride, as this reagent has the property of preventing the precipitation of mercurous chloride.

In the Indian Mints a humid method of assay is used in which the silver precipitated from solution by hydrochloric acid is washed, decanted into a porcelain capsule, dried at 200° C., and weighed as chloride.

## METALLURGY OF SILVER.

The greater part of the silver production of the world is, as has been previously stated, derived from ores containing only a very small proportion of that metal. In most of the larger mining districts, ores considered as first class rarely yield above 1 per cent. of silver, or 320 ounces per ton, and the bulk of the produce is considerably poorer. It will therefore be readily understood that the reduction of silver from the ore by a single operation is only possible in very exceptional cases, and that, as a rule, methods of gradual concentration must be adopted leading to the production of a small bulk of some highly argentiferous compound or alloy from which the silver may be finally extracted.

The substances in which silver is generally collected are the following:—

1. Metallic sulphides, particularly those of iron and copper; the *spurstein* or argentiferous *fine metal* of the Mansfeld process (page 439) is an example.

2. Metallic lead, producing the silver-lead of the ordinary lead-smelting processes.

3. Metallic mercury, producing amalgams of silver and other metals.

The methods adopted for the treatment of these enriched products constituting the characteristic operations of silver metallurgy proper are:—

For lead. Refining on the test or marl bottom as described under lead smelting.

For amalgam. Distillation or retorting, which is one of the subsidiary operations of the amalgamation process.

For argentiferous regulus. The production of some soluble salt of silver, which is removed by an appropriate solvent and reduced by cementation with copper, the series of operations being known as liquid extraction or leaching processes.

In the choice of a method for treating silver ores many points have to be considered, as the character and resources of the locality may often influence the selection as much as, or more than, the nature and composition of the minerals. Fusion with lead is probably the best method, as giving the least loss of precious metals; but it necessitates the command of fuel and fluxes, as well as of lead ores, in some quantity. Argentiferous copper ores are more difficult to work than similar ores of lead; they are best treated by fusion up to the production of a concentrated regulus, and then by liquid extraction; but the latter processes involve rather delicate manipulation, and for successful working well arranged plant and abundant fuel and ores for mixing are necessary. Pure or moderately rich dry ores, without either copper or lead in notable proportion, may be best treated by amalgamation, which requires only a minimum of fuel and comparatively small quantities of accessory materials, the latter being principally salt and mercury; while the mechanical appliances need only be of the simplest possible character. These advantages more than counterbalance the waste of the most expensive reagent employed, namely, mercury, and the loss of silver by imperfect extraction, which, generally speaking, is larger than in smelting, and sufficiently account for the circumstance that very much more silver is produced by amalgamation than by any other method of reduction.

### SMELTING SILVER ORES.

The various processes described for smelting lead ores are applicable to those of silver. When, however, the extraction of silver is the chief object in view, some form of blast-furnace is often employed, although the flowing furnace, used for the treatment of refractory ores, is also well adapted for the purpose. In all such cases the product first obtained is argentiferous lead, which may be either concentrated before refining, or passed directly to the test. Where silver ores have to be worked in a district in which fuel is moderately abundant, but ores of lead are obtained with difficulty, it will generally be found advantageous to refine the argentiferous alloy without preliminary concentration, and to employ the resulting litharge, fume, &c., continuously in smelting fresh quantities of argentiferous material.

When silver is found associated with large quantities of pyrites or metallic sulphides, other than galena, the ores are frequently fused for a coarse matte previous to fusion with litharge or other plumbiferous material. The matte thus obtained is then either roasted and subsequently smelted with some material capable of affording lead, or is first concentrated by a second fusion, followed by another roasting. Nearly the whole of the silver originally present in the ores treated will thus be concentrated in the mattes in a form convenient for subsequent metallurgical treatment.

The presence of gold in an ore of silver in no way modifies its treatment, since both metals invariably pass into the lead, and are ultimately separated, after refining on the test, by the process of "parting."

SMELTING NATIVE SILVER.—At Kongsberg, in Norway, about 80,000 ounces of silver are produced annually from ores that consist chiefly of native silver with some silver glance, and small quantities of copper-pyrites, galena, and blende in a matrix of calcite and quartz. These are divided into hand-picked and dressed ores which are treated separately. The first and most important class, consisting of lumps of mineral, often assaying up to 90 per cent. of silver, is melted in a small reverberatory furnace with an addition of lead and wrought-iron nails for decomposing the sulphide of silver, and produces highly argentiferous lead which is refined to pure silver in the same operation, while the dross and skimmings removed from the surface are added to the furnace charges in smelting the dressed ores, which are divided into richer and poorer classes, the first with about 300 ounces, and the second with 10 ounces of silver per ton. The poorer ores, mixed with flue-dust and other residues, tap-cinder and iron pyrites, are concentrated by fusion in a blast-furnace 14½ feet high, to a coarse metal containing about 100 ounces per ton, which is burnt in heaps four times, and concentrated by fusion with the richer dressed ores, flue stuff, &c. This enriched regulus is desilverized by running it into a bath of melted lead in the fore-hearth of the furnace, where the silver is reduced from sulphide and dissolves in the excess of lead, with the production of an equivalent of lead regulus or Bleistein. This process, known as *soaking*, is repeated twice with the same lead, which takes up about 10 per cent. of silver before it is sent to the refinery. The lead regulus, like the similar product of the Harz iron-reduction process (p. 609), is subjected to alternate roasting and fusion until it is deprived of its lead and silver and becomes copper regulus. When a quantity of about 10 tons of the latter substance has accumulated, it is smelted for blister copper and refined. The refined copper, containing from 1 to 1¼ per cent. of silver, is sold to a refinery to be used in precipitating silver. About 30,000 ounces of silver are obtained annually from the poorer ores by the treatment described above.

Rich dry ores, with 1,000 ounces of silver per ton and upwards, are sometimes treated by a soaking process, known as working on the test, in the following manner. When the production of litharge begins in the refinery the blast is stopped and the ore is spread over the surface of the melted lead, where it is slowly decomposed, and the reduced silver is absorbed by the lead in the same manner as in scorification in the muffle. Sulphur, antimony, and arsenic are oxidized and volatilized in part, the remainder forming with the earthy matters of the ore and some litharge a pasty mass, which is skimmed from the surface. This dross contains a notable quantity of silver and is worked with poorer ores in a blast- or flowing-furnace. The addition of ore continues until the lead is enriched to 4000 ounces, when it is tapped and refined on a clean test.

## TREATMENT OF SILVER ORES BY AMALGAMATION.

The amalgamation of silver ores is conducted in various ways, the following being the most important: Firstly, the *Mexican process* of amalgamation in heaps; secondly, the *barrel process*, formerly employed at Freiberg in Saxony, at Constante in Spain, in the State of Nevada, and elsewhere; thirdly, the *Washoe process*, by which unroasted ores are amalgamated in iron pans.

**MEXICAN OR PATIO PROCESS.**—This method of extracting silver from its ores by the use of mercury and common salt, without the assistance of artificial heat, was discovered in the year 1557 by Bartolomé Medina, a native of the town of Pachuca, in Mexico, and has been uninterruptedly employed from that period up to the present time, without having undergone any material modification.

The silver in the ores operated on chiefly exists either in the metallic state, or combined with sulphur, chlorine, iodine, or bromine. The arsenic and antimony contained in ruby- and brittle-silver ores not only render their treatment difficult and expensive, but also materially augment the loss of both silver and mercury. When sufficiently rich, such ores are often set aside for treatment by smelting, but they are generally so disseminated throughout the veinstone, and so intimately mixed with it, as not to admit of separation by any system of hand-picking. The gangue in which they are contained consists principally of quartz, more or less associated with pyrites or with iron oxide, and, in addition to small quantities of other minerals, often contains from 5 to 10 per cent. of calcite or dolomite. Near the surface such veins are much decomposed, and the ores then present the appearance of a friable ferruginous quartz, in which a large proportion of the silver occurs either in the native state or as chloride.

The ores treated by this process are not ordinarily subjected to mechanical preparation, as not only are they unsuited for such treatment, but the supply of water in the districts in which they occur is, in most cases, so limited as to render washing operations on an extensive scale impossible.

Although the loss of silver by the *patio* system of amalgamation is large, and much time is expended on the various operations, it nevertheless possesses advantages over all others, in the arid barren districts in which it is carried on, inasmuch as it requires only a little fuel, and no great amount of water. The principal materials necessary are salt, cupriferous pyrites, and mercury.

*Rough Stamping.*—The ores to be treated by patio amalgamation are first crushed dry in a *molino* or stamping-mill, and subsequently ground with water in the *arrastra* (or *arrastre*) until reduced to the necessary state of fine division. The stamping-mill generally consists, in Mexico, of a series of wooden lifters or stems, shod with iron, and set in motion by cams arranged round an axle, worked either by a water-wheel or by a vertical shaft, carrying a beam, to which is harnessed a team of four mules. The vertical shaft is provided with a large wooden wheel, which communicates its motion to another, fixed on the cam-shaft of the mill. Ore is supplied in pieces of the size of the fist, which, when sufficiently reduced in size, fall through sieves or screens, made either of metal or of raw hide perforated with numerous round holes, and fixed in an inclined position before each battery. The particles of ore which pass through these holes are removed for fine grinding in the arrastra, while the coarser portions continue to be acted upon by the pestles, until sufficiently reduced in size to admit of their passing through the screens. When animal power is employed these mills are worked by relays of mules, driven at a rapid rate, and frequently replaced by fresh teams.

*Fine Grinding.*—The *arrastra*, or *tahona*, consists of a circular pavement of stone, which may be about 12 feet in diameter, on which the ore is ground by two or more stone mullers dragged continually over its surface by mules harnessed to a horizontal arm. Around this circular pavement of hard stone is a kerbing, either of flat stones or of wood, forming a kind of tub, about 2 feet in depth, in the centre of which a piece of hard wood is firmly fitted between the blocks composing the flooring. Working on an iron pivot, in a step let into this central post, is an upright wooden shaft, secured at its upper extremity to a horizontal beam by another journal. This is crossed at right angles by two bars forming four arms, one of which is sufficiently long to admit of two mules being harnessed to it abreast. The *voladoras*, or mullers, are generally made either of porphyry or of granite, although basalt is also employed for this purpose, and have a length of somewhat less than the radius of the arrastra, with a thickness of about 16 inches. In each of these are bored two holes, into which wooden pegs are driven for attaching the chains, or thongs of raw-hide, by which they are connected with the arms traversing the central shaft. These mullers are so hung that their edges, in the direction of their motion, are raised nearly an inch above the surface of the stone pavement, while the other side trails heavily upon it. A sectional view of a Mexican arrastra is given in fig. 204, in which A is the upright shaft, B, arms to which the mullers, C (of which only one is shown), are attached, and D the central block of wood in which the lower bearing works.

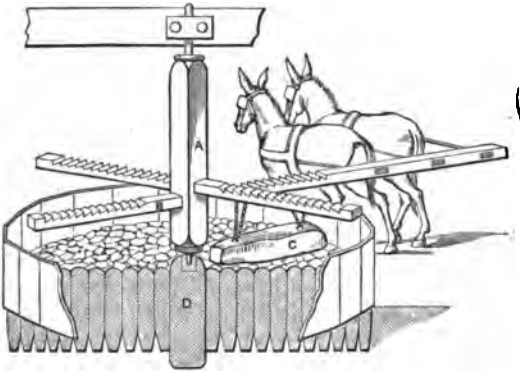

Fig. 204.—Arrastra; partly in section.

At Guanaxuato, where the ores in addition to silver contain gold, and are more finely ground than in other districts of Mexico, from 6 to 11 quintals[1] of *granza*, or coarse sand from the stamping-mills, are charged into the arrastra with one barrel, or about 10 gallons, of water, at four o'clock in the morning; at nine o'clock a second barrel is poured in, and a third at twelve; at three o'clock three barrels, and at four, five barrels, of water, are added. At the expiration of twenty-four hours, when the grinding has been completed, the *lama*, or argentiferous mud, is baled out into barrels, in which it is removed to reservoirs of masonry, where a portion of the water is evaporated by exposure to the sun and air, and the mass is ultimately left in a condition fit for treatment in the patio. In some establishments, instead of the lama being taken away in barrels it is baled into wooden spouts, by which it is conducted to proper receptacles; while in others it is tapped from a plug-hole at the bottom of each arrastra directly into properly arranged spouts. The mules by which the machines are driven are changed after working six hours, and the arrastras are generally arranged in rows in long sheds called *galeras*. At Zacatecas, where the ores contain no gold, the grinding is not so long continued, and the lama is removed in a less finely divided state. In this district each arrastra grinds 10 quintals in the course of thirteen hours, but the lama is much coarser than at Guanaxuato, and the results obtained less satisfactory.

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[1] The Mexican quintal = 100 lbs. avoirdupois, nearly.

Where, as at Guanaxuato, the ores contain, in addition to silver, small quantities of gold, the arrastra is kept constantly charged either with a certain amount of mercury or with an amalgam of silver or copper. By this means the gold becomes concentrated in the amalgam, and highly profitable results are sometimes realized. In such cases, care must be taken, in removing the lama daily obtained, not to disturb the amalgam in the bottom of the arrastras, which is collected at periods of from three to six months, and, after being strained and retorted, is melted into bars which are subsequently subjected to parting.

The yield of gold by this method is considerably less than the total amount contained in the ores, the loss experienced generally varying from 25 to 40 per cent. on the assay produce. A loss of mercury also takes place during the process, which is apparently caused by the decomposition of sulphide of silver with the formation of a proportionate amount of sulphide of mercury. The loss of this metal is usually found to represent very closely the weight of silver taken up by the amalgam during the progress of the operation.

*The Patio.*—This is a large courtyard, paved with stone, the joints between which are carefully cemented in order to prevent loss of mercury. A slight inclination is given to the surface of the floor, in order that any water that may fall upon it may readily flow away. The ground ores are, as before stated, taken from the tahonas to walled receivers (*lameros*), in which they become partially dried, and where they are allowed to accumulate until there is a sufficient quantity to form a heap, or *torta*, which, at Guanaxuato, consists of 60 *montones*, about 96 tons.[1] When the necessary amount of lama has been thus collected it is taken to a circular area on the patio, from 30 to 50 feet in diameter, according to the weight of ore operated on, surrounded either by a low wall of stone or by a border of planks made tight by filling all the crevices with clay or mule dung. Into this the lama is introduced until it forms a stratum of about a foot in thickness, and is allowed to remain until, by the evaporation of the water, it has assumed the consistence of a moderately thin mud. As soon as this condition has been reached, the amalgamator proceeds to add from 3 to 5 per cent. of salt, and when this has been done the torta receives the first treading (*repaso*), after which it is allowed to stand until the following day. A larger amount of salt would, in many cases, expedite the working of the torta, but on account of its high price this is seldom added.

In addition to common salt, imported from the coast, a large quantity of impure sodium chloride, obtained from various lagunes, was formerly employed, but the cost of transporting considerable amounts of such an impure material, added to the great increase in bulk of the tortas, has caused its use, in a crude state, to be almost entirely abandoned. This salt is now concentrated and purified by lixiviation and evaporation before being carried to the mines.

The day after salt has been mixed with the ore, the addition of mercury and *magistral* takes place. Magistral is prepared by slowly roasting, in a reverberatory furnace, copper pyrites containing a considerable admixture of iron pyrites, to which a small quantity of common salt has been added. By this process, the minerals present in the raw ore become oxidized with the formation of cupric and ferrous sulphates, together with a small proportion of chlorides. The cupric sulphate varies in amount from 20 to 40 per cent. of the whole, and is the chief agent in the reduction of the ores, although the iron salts, which are present in amounts varying from 6 to 12 per cent., also exercise a beneficial influence on the results.

Before the addition of magistral, the torta, if necessary, brought to a proper consistence by the addition of water, and the roasted sulphides are then thrown evenly over its surface with wooden shovels. The proportion of this reagent added is varied in accordance with the amount of cupric sulphate it contains; but in the generality of cases, when magistral of average strength is employed, something less than 1 per cent. is sufficient.

As soon as the magistral has been spread over the surface of the torta, it is again trodden by mules for about an hour, when the mercury necessary for the completion of the operation is added. Formerly, this

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[1] The weight of the monton varies in different localities—

| | | |
|---|---|---|
| In Guanaxuato a monton usually contains | 32 | quintals |
| ,, Real del Monte, Pachuca, and Tasco | 30 | ,, |
| ,, Zacatecas and Sombrerete | 20 | ,, |
| ,, Fresnillo | 18 | ,, |
| ,, Bolaños | 15 | ,, |

addition was made at various periods during the progress of the operation, but it is now more usual to add all the mercury immediately after the introduction of magistral. This is done by straining it through a piece of canvas, by which its particles are divided into minute globules, the quantity added being from 3½ to 4 lbs. for each mark,[1] or from seven to eight times the weight, of silver supposed to be contained in the heap. After the addition of mercury the torta is again trodden, to effect its intimate mixture throughout the mass.

When magistral and mercury have been added to a torta, and it has received its first treading, chemical action commences, and the amalgamator closely watches its operations by means of samples taken from all parts of the heap. To make an assay or *tentadura*, a fair sample of about 8 ounces of the ore is washed with water in a small bowl (*jicara*), and from the results obtained the *azoguero* (amalgamator) is enabled to judge of the progress of the operation. Shortly after the first treating of the torta samples are taken, and a tentadura is made, and, after washing off the earthy and lighter particles, the remaining *polvillos*, or metallic sulphides and mercury, are carefully examined. At this stage the mercury contains but little silver, and its colour and state of division afford the only indications of the working of the torta. Should it be found divided into small globules, or its natural colour be but slightly changed, it indicates that the amount of magistral added is not sufficient. If, on the contrary, the mercury has acquired a deep grey or leaden hue, the quantity of magistral is too large, and the torta is said to be *hot*, in which case the addition of a small quantity of lime may be necessary in order to avoid undue loss of quicksilver. When the heap is in good working order, the surface of the mercury presents a light-grey appearance, and the aspect of the tentadura, taken the day after treading, will have considerably changed. If now pressed by the thumb against the side of the bowl, the mercury will be found to contain silver amalgam, and what on the previous day was broken-up quicksilver (*desecho*), has become *limadura de plata*, of a whitish colour and in the form of thin scales. Three tentaduras are made daily on each torta: one in the morning before commencing to tread, another after it has been trodden for some time, and a third when the repaso has been completed. The samples selected for this purpose must be taken from the middle of the heap, as well as from the surface, since the top, from being exposed to the action of the air and sun, is always in a more advanced condition than the interior. The treading, which must be repeated daily, or as often as tentaduras indicate a necessity for doing so, has the effect of stimulating the action of the magistral. When chemical action has almost ceased, and nearly the whole of the silver which the process is capable of extracting has been taken up by mercury, the limadura becomes " weak," and on being rubbed by the thumb shows but little solid amalgam. As soon as it is found to be free from amalgam, and unites

[1] Mark = 3,550·5 grains.

in globules at the bottom of the jicara, the operation is considered finished, and the torta is said to be *rendido.* For many years, however, amalgamators have not entirely relied on the results obtained by washing, but have also been assisted by fire assays made on average samples taken from each torta; in this way its content of silver is more exactly ascertained. Another weighed sample is washed in the jicara, and the mercury and amalgam carefully collected; the assays of these afford data for calculating the proportion of silver which still remains unacted upon in the ores. The treading is performed by mules, which are blindfolded and tied together four abreast. One mule for every two montones of ore is, at Guanaxuato, required for the effectual treading of a heap. A driver, who stands in the centre of the torta, guides the animals with a long halter, causing them first to tread at the outer edge, and gradually diminishing the radius of the circle described. The time necessary for working a torta varies from fifteen to forty-five days, according to circumstances. When the working of a heap has been completed, a quantity of fresh mercury, called a *baño*, is sometimes added, but this practice is not universal.

*Washing.*—At Guanaxuato the washing apparatus (*lavadero*) consists of three circular tanks built closely together within an outer circle; these are constructed of stone slabs with carefully cemented joints. The depth of each tank is 5 feet 4 inches, and its diameter 9 feet 6 inches; they communicate with one another by openings, of which one is at a height of 8 inches and the other at a distance of 30 inches from the bottom. The last tank is provided with two separate discharge holes; the first 6 inches from the bottom, and the other, which is used only for cleaning-up, is situated close to it.

In the centre of each tank is an upright shaft carrying agitating arms; the whole being set in motion by a shaft provided with a spur-wheel working in pinions on the shafts of the different washing-vats. The shaft carrying the spur-wheel passes through an upper flooring, where motion is communicated to it by a team of mules attached to an arm let into it at right angles; the pinions of the agitators in the second and third tanks are a little larger than that working the stirrer in the first, and their motion is consequently somewhat slower. The first tank into which the metalliferous mud from the torta is charged is called the *tina cargadora*, whilst the third, from which it runs off, after passing through the second, is called the *descargadora.* Before being washed, the torta is divided into several parcels, each of which is softened by treading and the addition of water, and then taken to the washing-house in *bateas*, dusted on the inside with dried mule dung to prevent adherence. About three montones of lama are gradually introduced into the first tank, water being at the same time run in, and the machinery is made to revolve rapidly, by driving the mules at a gallop. When the whole of the lama has been introduced, the speed of the agitator is gradually slackened, until the mules move only at a gentle walk, and samples of the slime are, from time to time, taken out and washed, in order to ascertain if it still retains an appreciable amount of mercury. When samples taken from the tinas afford only minute traces of metal, the plug furthest from the bottom of the descargadora is removed, and, as soon as the slimes have run off, they are replaced, and the operation is continued until the whole of the torta has been washed. The bottom plug is then removed with suitable precautions, and the whole of the mercury and amalgam are discharged and carefully collected.

At Guanaxuato, the heavier constituents of the torta, which remain with the amalgam at the bottom of the tinas, are separated from the latter by washing in bateas, and the resulting *relaves* are subsequently re-ground in arrastras. By this treatment they are made to yield a certain amount of auriferous amalgam, but they are not always again subjected to patio amalgamation. In this district the slimes from the lavadero are now sometimes concentrated by the round buddle.

At Zacatecas and Fresnillo the washing is conducted in a single circular cistern, and as soon as the azoguero considers that a torta is ready he adds to it about 80 per cent. of the amount of mercury it already contains. At Guanaxuato the lama is taken direct to the lavadero without any further addition of quicksilver. The speed of the agitator is greater than in cases where three tinas are employed, and about two and a half montones of lama are passed through each cistern in the course of an hour. A considerable loss of amalgam is the result of this system of washing, and in order to recover it, the heavy residues collected in a cistern beneath the discharge-orifice of the tina are re-washed on a *planilla*, or washing-table. The concentrates from the washing, when sufficiently rich, after being previously roasted in a reverberatory furnace, are re-ground in an arrastra, and a second time treated by amalgamation in the patio. In certain districts, and particularly in those situated nearest to the city of Mexico, the washing of the tortas is conducted in a wooden tank, of which one end is pierced with numerous holes at different heights, which admit of being closed by plugs provided for that purpose. At the opposite end a stream is admitted, and as soon as the cistern has become nearly filled with water the lama to be washed is thrown in and briskly stirred with shovels; when the ore is thus well incorporated with water the plugs are successively removed, beginning with the upper one. The lighter earthy matters are in this way first drawn off, and afterwards the heavier metallic sulphides, until the amalgam, in a tolerably pure state, remains in the bottom. After escaping from the tank, the slimes are conducted through riffle-boxes for a distance of from 70 to 80 feet, in which a considerable portion of the suspended amalgam and mercury is retained.

*Filtration of Amalgam.*—The amalgam obtained is first purified from adhering particles of mineral, and then filtered through a cone-shaped bag (*manga*), of which the upper portion is cased with leather, while the lower part consists of stout canvas only. This is hung, point downwards, from a strong beam, and the mixture of mercury and amalgam introduced. The former gradually passes through the meshes of the canvas, and is collected in a vessel placed beneath for that purpose. The amalgam remaining in the manga contains mercury to the extent of from four to five and a half times the weight of silver present, and has a granular and plastic consistency, which readily admits of its being moulded into bricks. As soon as mercury has ceased to drop from the point, the manga is emptied on a table covered with leather, and the amalgam is beaten into bricks (*bollos*), from 3 to 4 inches in thickness, and so shaped, that when six of them are placed together they form a circular cake, having a hole in the centre for the escape of mercurial vapours, during the subsequent process of distillation.

*Retorting.*—The separation of mercury from the silver is effected by distillation under a large cast-iron bell placed over the amalgam, and around which is lighted a charcoal fire. A circular casting, through which a current of cold water constantly circulates, is let into the floor of the retorting house, and on this is placed an iron support, on which the bricks of amalgam are arranged. When thus prepared the bell (*capellina*) is lowered over it, and the bottom carefully secured by a luting of clay. Unburnt bricks (*adobes*) are now built around it in the form of a hollow wall, so as to enclose an annular fire-place, 8 inches in width, outside the capellina; this is filled with ignited charcoal, and in proportion as the mercury becomes volatilized, it is condensed by the action of the cold water, and escapes through an iron pipe into a receptacle prepared for that purpose. Instead of the wall of adobes an iron cage is sometimes placed around the bell, for the purpose of retaining the charcoal. The resulting silver (*plata pina*) has a porous frosted appearance. The spongy silver thus obtained is fused and cast into bars in the ordinary way, and is generally above 990 fine.

*Results obtained, &c.*—The loss of silver experienced in patio amalgamation is always considerable, but varies in accordance with the nature of the ores and the amount of skill brought to bear upon their treatment. At Guanaxuato the average loss on docile ore is from 9 to 14 per cent. on the assay produce. At Fresnillo the deficit is often 28 per cent., whilst according to Duport, the loss at Zacatecas, where the ores contain a large proportion of antimonial sulphides, was, at the time he wrote (1843), from 35 to 40 per cent. of the yield indicated by assay. The loss of silver in patio amalgamation at Real del Monte (1864–65), as furnished by Mr. Buchan, was only 9 per cent.

The loss of mercury experienced at the same period in the latter establishment was 11 ounces per mark of silver; but this may be taken as an unusually favourable result, and the average may probably be estimated at from 12 to 16 ounces per mark of silver extracted.

The cost of patio amalgamation varies in different localities; at the Hacienda Nueva, belonging to the Fresnillo Company, the cost of treating 2,000 lbs. (1840–41) was $20·74, while at the Ophir Company's works, Nevada (1864), the cost of working the same weight of ore was $23·25.

The chemical reactions which take place during the progress of patio amalgamation are of a complicated nature, and the whole of them are not yet thoroughly understood. The ore contains a mixture of native silver, chloride of silver, and various sulphides, &c., containing silver. Common salt is decomposed by cupric sulphate in the magistral, giving rise to sodium sulphate and copper and iron chlorides. Cupric chloride in its turn reacts on silver sulphide with the production of silver chloride, which is dissolved in the excess of sodium chloride added to the torta, and the silver is subsequently reduced to the metallic state by a portion of the mercury, which is ultimately converted into calomel, while the reduced silver forms an amalgam with the residual unaltered mercury. The cuprous chloride formed is dissolved in the excess of sodium chloride, and converts another portion of silver sulphide into chloride, which is subsequently reduced by mercury, and finally forms an amalgam with that metal. The copper is, ultimately, chiefly converted into sulphide, and mercuric sulphide is sometimes also found in the torta; this has been supposed to be the result of the action of mercurous chloride on silver sulphide. It has, however, been shown that this substance may be produced by the direct decomposition of silver sulphide by metallic mercury, and it is highly probable that this reaction, to a certain extent, takes place in the torta. It has been contended, by persons practically acquainted with patio amalgamation, that silver chloride is not necessarily formed during the process; but the various phenomena brought forward in support of this view appear to be capable of being otherwise interpreted.

STOVE AMALGAMATION.—In some of the colder and more humid districts of Mexico a modification of patio amalgamation has been sometimes resorted to. The ground ore from the arrastra is placed in a shed, where the salt, magistral, and mercury are added, and the process is conducted in the usual way. When the operation is about half completed it is removed to a stove (*estufa*), consisting of a chamber with flues, so arranged beneath the floor as to communicate to the mixture the heat of a fire-place with which they are connected. It is here exposed to a gentle heat during from two to three days, and is then taken back to the shed, where the operation is completed by the ordinary method of patio amalgamation. In this way the time necessary for the reduction of the ore is diminished, and the yield of silver somewhat augmented; the loss of mercury is, on the other hand, more considerable.

HOT PROCESS.—In parts of South America where the ores contain a large amount of native silver, or where that metal occurs in combination with chlorine, iodine, or bromine, amalgamation is often effected by the *cazo*. This apparatus consists of a vessel formed either of blocks of stone or of wooden staves, like those of a tub, the bottom of which is made of a slab of copper 2½ inches in thickness, which becomes gradually thinned by use. This metallic bottom is retained in its place by a groove running round the interior of the vessel, in the same way that the head of a cask is secured, and all joints are carefully luted with clay. The copper plate rests upon the walls of the hearth, forming the roof of a fire-place, which has neither fire-bars nor chimney, and which has but one opening for both the introduction of fuel and the egress of smoke. After being roughly stamped, ores intended for cazo amalgamation are ground in the arrastra; but as they are subsequently subjected to a process of washing, care is taken not to carry the operation too far. The ordinary dimensions of the cazo are: diameter at top, 3 feet 3 inches; at bottom, 2 feet; depth, 18 inches. About 100 lbs. of concentrated ore forms the charge, in which it is mixed with as much water as will convert it into a paste. When the contents of the vessel have been made to boil, from 5 to 10 per cent. of salt is added, and the workman, who squats by the side of the furnace and keeps the mixture constantly agitated with a wooden stirrer, begins the addition of mercury. More mercury is from time to time added, and numerous samples are taken and washed in order to ascertain to what extent the ores have become exhausted of their silver. The total amount of mercury added is usually twice the weight of the silver supposed to be contained in the charge, and the duration of the entire operation is six hours; the liquid contents of the cazo, together with the slimes, are now dipped out into reservoirs, whence they are subsequently removed for further treatment by patio amalgamation. By this process the silver which exists in the ores in a native state, as well as that combined with chlorine, iodine, or bromine, is readily made to unite with mercury, but silver sulphide does not easily yield its metal to cazo amalgamation, and hence the necessity of re-treating the residues in the torta. They do not, however, require the addition of magistral, since they contain sufficient chloride of copper to convert the whole of the silver sulphide into chloride.

In the district of Catorce, Mexico, the dimensions of the cazo are sometimes much enlarged, and, under the name of *fondon*, this contrivance is somewhat extensively employed for the reduction of argentiferous ores. The diameter of the copper bottom of the fondon is from 5 feet 6 inches to 7 feet 6 inches, and, instead of the necessary motion being imparted to the charge by a wooden stirrer, worked by hand, it is obtained by blocks of copper dragged over the bottom by an arrangement similar to the arms of an arrastra. It may, therefore, be regarded as an arrastra in which the stone paving and voladoras are replaced by a plate and blocks of metallic copper, to which a mule is harnessed in the ordinary way; a fire-place is built below the bottom, which is provided with a plug-hole for tapping off the slimes at the termination of each operation. The charge varies from 1,200 to 1,500 lbs., and its treatment is completed in six hours, as in the case of cazo amalgamation. The loss of mercury experienced during the treatment of silver ores in the cazo and fondon is extremely small, not amounting to more than 2 to 3 per cent., since by this process chloride of silver is finally reduced at the expense of the copper bottom of the apparatus, instead of by the action of mercury, as in patio amalgamation.

**Barrel Amalgamation.**—The amalgamation of silver ores by this process was conducted with great skill and economy at Halsbrücke, near Freiberg, from the year 1790 up to 1856, when it was finally abandoned on account of the expense of manipulation and of the unsatisfactory results it afforded when applied to certain classes of ore. The changes of more than half a century had also so modified the relations originally existing between the prices of labour, fuel, &c., that, at the latter date, it was found advantageous to abandon the use of mercury, and to smelt the argentiferous ores with others containing lead. The usual constituents of the ores treated at Halsbrücke are sulphur, antimony, arsenic, silver, copper, lead, iron, and zinc, which are more or less mixed with various earthy minerals, besides sometimes containing small quantities of bismuth, gold, nickel, and cobalt. In the selection of these ores they were so assorted as not to contain above 4 per cent. of lead, or 1 per cent. of copper, as these metals, when combined with the mercury, give to the amalgam a pasty consistency, thereby rendering the treatment both difficult and expensive, besides lowering the fineness of the retorted bullion.

The different ores selected for amalgamation varied in richness from 15 to 200 ounces per ton. At one period the mixtures of these ores were so arranged that the charges should always contain from 75 to 80 ounces per ton. Latterly, however, it became usual to work the poor and rich ores separately, since it was found that the total loss of silver in the residues was thereby diminished.

The mixtures of the poorer ores contained, on an average, from 30 to 40 ounces per ton, and those of the richer ones from 90 to 130 ounces of silver per ton. It is essential that both mixtures should contain a certain minimum proportion (about 25 per cent.) of sulphide of iron, for the formation of iron sulphate, which is necessary to the success of the roasting process. If the amount of iron pyrites, naturally occurring in the ores, did not equal this proportion, addition was made either of that mineral or of ferrous sulphate. Frequently, however, the ores at Freiberg contained more pyrites than was required, and in such cases some of the more sulphurous varieties were subjected to a previous roasting without salt, in order to reduce the amount of sulphur in the whole to the right proportion.

The ore when prepared was laid on a floor, 40 feet in length and 12 in width, and on the top of it was thrown 10 per cent. of common salt, which was let down from another room, through spouts placed in the floor. The heap, when it had been thus made up of alternate strata of ore and salt, was well mixed by being turned over with a shovel, and then passed through a coarse sieve. It was subsequently divided into small parcels, each weighing from 4½ to 5 cwts. The salt annually employed for this purpose at the Halsbrücke works amounted to 500 tons, and was supplied by the Prussian salt mines.

The mixture of ore and salt was then roasted in reverberatory furnaces, provided with long chambered flues for the reception of the pulverulent matters taken over by the draught.

The prepared charge was spread on the bottom of the hearth, where it was at first gently heated, for the purpose of expelling the moisture which it invariably contained. During the drying, which occupied from twenty to thirty minutes, the charge was continuously stirred by a long iron rake. The lumps, which were formed by this operation, were then broken down by an iron beater, provided with a long handle. The heat was afterwards raised, white fumes were given off, and, in about two hours from the commencement, the mass had become red hot. The charge was occasionally turned, so that every particle of ore might be equally exposed to the fire, and during the whole time the mass was diligently stirred. The fire was now allowed to burn down, and the oxidation of the sulphur aided by constant stirring. This went on without intermission until the mass became quite dark, and a sample taken from the furnace no longer evolved any sulphurous odour. During this period the ore increased in volume, and the particles hung so loosely together that the movement of the rake was scarcely at all impeded. The heat was again raised for three-quarters of an hour; the iron sulphate, formed by the oxidation of pyrites, reacted on common salt, and caused the evolution of chlorine and hydrochloric acid gases, which, coming in contact with various silver compounds, converted them into chloride. Chlorides of the other metals present were at the same time formed, together with sodium sulphate. When the roasting was terminated, the charge was raked from the furnace into an iron barrow, and thence removed to an adjoining floor. The ore was afterwards raised to an upper story for the purpose of being passed through a set of sieves, by which the finer powder was separated from the agglutinated lumps. These were broken to a proper size, and a portion re-roasted by adding a small quantity to each of the ordinary charges. The remainder was mixed with 2 or 3 per cent. of salt, and calcined in the usual way. The finer particles, which passed through the sieves, were taken to heavy millstones and reduced to the state of an impalpable powder.

After roasting, the charge consisted chiefly of oxide of iron, basic sulphate of iron, and protochlorides and perchlorides of iron and copper; together with oxide and sulphate of copper, sulphate of lead, oxide of antimony and zinc, and a small quantity of metallic sulphides, in addition to gangue, various earthy salts, sulphate of sodium, and an excess of common salt. The compounds of silver, originally present in the mineral, were converted into chloride, with the exception of traces of metallic silver, and perhaps also of a minute quantity of sulphide of silver, which remained in the residues. The charge in roasting suffered a diminution in weight, amounting to about 10 per cent., due to the escape of sulphur, chlorine, particles of salt, zinc, antimony, arsenic, chloride of iron, &c.

The amalgamation of the roasted ores was performed in twenty wooden casks, arranged in four rows, each turning on iron gudgeons, secured to their ends by iron bolts. These barrels, which were internally 2 feet 8 inches in length, 2 feet 8 inches in diameter at the ends, and 2 feet 10 inches in the middle, were made of pine, 3½ inches in thickness, and strengthened by iron hoops and binders, fig. 205. On one of the ends of each barrel was a oothed-wheel, *w*, figs. 206, 207, communicating with a shaft, *x*, which received its motion directly from a water-wheel.

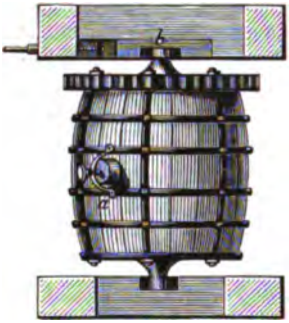

Fig. 205.—Amalgamating Barrel.

Above each of the barrels was a wooden case, C, into which was thrown the prepared charge. To the bottom of this case was fixed a wooden spout, to which was attached a hose made of strong cloth, and terminated by a short cylinder of tin-plate, for introducing the charge into the barrel, B. Each cask was furnished with a circular opening, *a* (figs. 205, 206), 5 inches in diameter, fitted with a wooden plug, through which had been bored a small hole, for running off the argentiferous mercury at the termination of the process, provided with a pin made of hard wood. Below the barrels, and a little above the surface of the floor, were triangular troughs destined to receive the residual matters at the close of the operation. At the commencement of the process 3 cwts. of water were run into each barrel, after which 10 cwts. of the roasted,

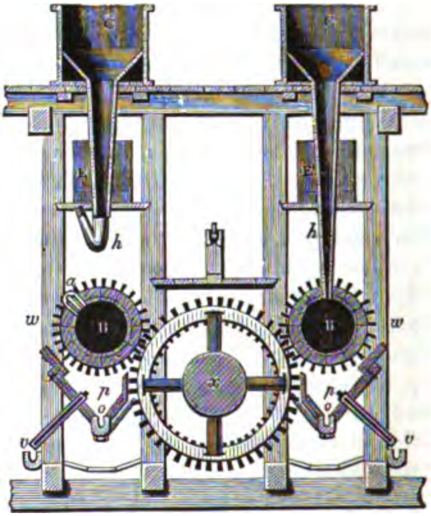

Fig. 206.—Amalgamating Barrels; transverse section.

ground, and sifted ores were introduced through the hose, *h*. Each cask contained from 80 to 100 lbs. of wrought-iron, cut into fragments about 1 inch square and $\frac{3}{8}$ of an inch in thickness, which, in proportion as they became dissolved, were replaced by fresh pieces.

As soon as the barrels were charged, and the plugs secured in their places by binding screws, they were thrown into gear with the driving-wheel by a screw and sliding block, *b* (fig. 205), and made to rotate at from twelve to fifteen revolutions per minute.

At the expiration of two hours the machinery was stopped, for the purpose of examining the state of the metalliferous paste. If the charge was too stiff a little water was added, but if, on the contrary, it was found to be too soft, a small quantity of ore was thrown in. When this had been attended to, 5 cwts. of mercury were poured into each cask, and the barrel, after being securely closed, was again thrown into gear, and kept constantly revolving for about sixteen hours, at the uniform rate of thirteen turns per minute. During the first eight hours of this period they were twice examined for the purpose of seeing whether the paste was of the proper consistence, for if too thick the mercury became too finely divided, and if too thin, it remained at the bottom of the cask, and was not sufficiently mixed with the constituents of the charge. In the first case, it was necessary to add water; and in the second, a little powdered ore. After the introduction of the mercury the temperature of the casks was raised considerably by the chemical changes going on within, so that even in winter it sometimes stood as high as 40° C. At the expiration of eighteen hours the amalgamation was ordinarily complete, and the casks were then entirely filled with water, and again made to turn during from one and a half to two hours with a velocity of only six or eight revolutions per minute. The mercury was thus separated from the slimes with which it was mixed, and collected in one mass at the bottom of the barrels. When this union of the globules of mercury had been accomplished, the different casks were successively thrown out of gear and stopped with their apertures uppermost. The small peg in the bung was then removed, and in its place was inserted a

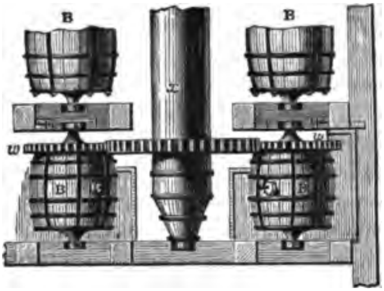

Fig. 207.—Amalgamating Barrels; seen from above.

hollow plug, to which was attached a leathern hose, with a screw-clasp for choking it when required. The cask was then turned round so that the plug, *a* (fig. 206), was over the spout, *o*. The hose being put into the iron tube, *p*, the mercury was allowed to run off into the gutter, *v*, by which it was conducted to a receiver. The workman closely watched this period of the operation, and the moment any earthy matter began to flow, the orifice was again tightly closed. The casks were then turned with their apertures, *a*, upwards, the small hose-plug was removed, and the bung loosened.

The bung-holes were afterwards again turned downwards, the bungs withdrawn, and the muddy residuum discharged into the trough immediately under them, from which it flowed into large washing-vats placed below the level of the barrels.

During the first two hours the casks were in motion, and before the introduction of mercury, the perchlorides contained in the ore were reduced to the state of protochlorides, and the saline matters dissolved by the water present. If, instead of this, the mercury had been introduced into the casks, together with the ore, it would, by reacting on ferric chloride, &c., have become partially converted into calomel, resulting in a loss of that metal. This is, however, avoided by the action of metallic iron, as the resulting ferrous chloride is without action upon mercury.

The chloride of silver in the roasted ore is decomposed by agitation with the metallic iron and quicksilver; the chlorine combines with iron in the form of chloride of iron, whilst the silver is dissolved in the liquid mercury. The chlorides of lead and copper which may be present are also reduced at the same time as the chloride of silver, and those metals enter into the composition of the amalgam obtained.

The slimes, conducted to the washing-vats before mentioned, were mixed with an additional quantity of water, and kept constantly stirred by bars attached to iron arms fixed to an upright shaft in the centre of each vat, and turned by a water-wheel. These vats were furnished with openings at various distances from the bottom, by which the muddy water was successively drawn off into tanks, where the solid matters were allowed to settle. These, if they contained as much as 4½ ounces of silver to the ton, were removed to a drying floor, and subsequently re-roasted with 15 or 16 per cent. of pyrites and 5 or 6 per cent. of salt, and the roasted product, after sifting, but without being re-ground, was subjected to amalgamation in barrels for a somewhat shorter period than the original ores.

The quicksilver collected in the bottom of the washing-vats was drawn off every five or six weeks, and, on account of the large proportion of impurities it contained, was treated apart from the ordinary amalgam obtained by tapping the barrels when the working of the charge was finished. This was afterwards filtered through canvas bags, by which the liquid quicksilver was separated from the pasty amalgam retained by the closeness of the web, while the mercury passed through into suitable reservoirs. The amalgam which was collected in the bags consisted of a mixture of 6 parts of mercury and 1 of an alloy composed of about 80 per cent. of silver and 20 of a mixture of copper, antimony, zinc, lead, and other metals. This amalgam was subsequently heated in iron retorts, and the mercury separated by distillation from the non-volatile constituents, which were obtained in the solid form. The employment of retorts had latterly almost entirely superseded the iron bells at one time used at Freiberg for this purpose. Three retorts were employed, and in each were placed 450 lbs. of amalgam on iron plates, which yielded about 70 lbs. of sponge or plate silver, *Teller-silber*. The time required to complete the distillation was about ten hours. The silver thus obtained was alloyed with various other metals, which, with the exception of a certain proportion of copper, were removed by a process of refining.

The loss of silver by this process is stated to have been from 5 to 9 per cent. of the amount contained in the ore, and the mercury expended varied from one-third to one-fourth of the weight of silver produced. According to Winkler, the average loss of mercury at Freiberg during seven years was about 3 ounces per lb. troy of silver retained. At Constante, Spain, the loss of quicksilver was (1856) 8·9 ounces per lb. of silver, and the loss of the latter metal, on the assay produce, 12 per cent. At Real del Monte, Mexico (1864–65), the loss of mercury was 7·6 ounces per lb. of silver obtained, and the loss of silver 13 per cent. The loss of mercury at the Ophir Company's Works, Nevada (1867), was 2·96 ounces per lb. of bullion, and that of silver 10 per cent. on the assay produce.

The cost of barrel amalgamation necessarily varies in accordance with the nature of the ores treated, and with the locality in which the works are situated. At Constante the total cost of treating one ton of ore, containing on an average 100 ounces of silver, was (1855) £2, 5s. 6d., while at Real del Monte (1864–65) it amounted to about £3, 10s. per ton. The cost of treatment by barrel amalgamation at the Ophir Works, Nevada (1867), was $20·14 per ton of 2,000 lbs.

WASHOE PROCESS OF AMALGAMATION.—Shortly after the discovery of rich silver mines at Virginia City, Nevada (1859), it became evident that on account of the prevailing high prices of labour, fuel, forage, and all other necessaries, none of the processes employed in other localities for the treatment of silver ores could, in that district, afford profitable results, if applied to material assaying from $30 to $60 per ton, of which the Great Comstock lode was then capable of yielding vast quantities. Under such circumstances it was important that some means should be devised for extracting the silver from such ores, without the preliminary roasting required for the barrel process on the one hand, and without the great expenditure of mercury, time, and labour necessary for patio amalgamation on the other. In addition to the expense and other disadvantages of the latter process, it was found, after numerous trials conducted on a large scale, that the climate of Nevada materially interfered, during a large portion of the year, with the chemical reactions of the torta. The experiments undertaken with a view of overcoming these various difficulties, finally resulted in a system of amalgamation in iron pans, which, from having been first introduced in the Washoe district, is generally known as the "Washoe process."

The applicability of this method to the treatment of argentiferous ores depends, to a very great extent, on their composition, and the nature of the various minerals with which they may be associated. The veinstone from the Comstock mines chiefly consists of crumbling white quartz, with which is mixed a certain amount of clay and country rock. A closer inspection generally shows the presence of iron and copper pyrites, and a still more careful search, if aided by the use of a hand-lens, reveals the presence of blende, galena, and argentite. More rarely stephanite and polybasite may also be distinguished among the minerals. Specimens may be found in which many, or all, of these minerals can be distinctly recognised. Gold occurs in the ores from the Comstock vein to the amount of above one-third their total value.

For the purpose of metallurgical treatment they were formerly, and to a certain extent still are, divided into three classes. The first class embraces those ores whose assay value is above $150 per ton of 2,000 lbs.; the second class includes ores ranging from $90 to $150 per ton; and the third class comprehends all workable ores of lower value than the foregoing, the average assay value varying considerably in different mines. The silver of ores of the first class is so intimately associated with lead, antimony, arsenic, iron, and other base metals, as to render its extraction difficult, and they consequently cannot be profitably treated by the simple processes to which the more docile ores of the second and third classes are subjected. Ores of the first class are not, at present, of frequent occurrence, but formerly they were crushed dry, roasted with common salt in reverberatory furnaces, and amalgamated in barrels. The ores of the second and third classes are treated by the Washoe or pan process; the chief differences, where any exist, in the details of treatment of the two classes being in the time required for amalgamation, and in the amount of quicksilver and "chemicals" employed.

The ore treated by the Washoe process is raised from the mines in fragments of various dimensions, and before being subjected to amalgamation requires to be brought to a state of minute division. Blake's mechanical stone-breakers are employed for reducing them to a suitable size for feeding into stamping-mills, in which they are usually crushed wet to the state of fine sand, and thence pass off, in suspension in water, through iron screens perforated with small holes, and are collected in suitable reservoirs, from which they are subsequently removed for the purpose of being ground in cast-iron pans with hot water and mercury, with or without the addition of chemicals. The amalgam thus obtained is separated from redundant quicksilver by straining, and afterwards retorted, and the residual alloy melted into bars.

*Stamping-Mill.*—The stamping-mill consists of a series of heavy iron pestles, which are successively lifted to a height, varying from 9 to 12 inches, and allowed to fall with their full weight on the ore beneath them. These stampers are enclosed in a mortar or battery-box of cast-iron, which is kept constantly supplied with ore, from which it can only escape by passing through screens, the degree of fineness of the apertures in which regulates its state of division. The mortars are rectangular in form, and usually contain five stampers, forming what is called a "battery;" they are supported on a solid wooden foundation, and the whole machine is established within a substantial framework of timber. Motion is given to the stampers by a series of cams keyed on a cam-shaft placed immediately in front of the battery, which is made to revolve, either by water or steam power.

Fig. 208 is a front elevation of two five-stamp batteries, and fig. 209 a transverse section of the same machine, on the line A B. These drawings have been reduced from the work, published by the United States Government, on the 'Mining Industry of the Fortieth Parallel,'[1] under the direction of Mr. Clarence King. The stampers move vertically between guides of hard wood, G, *g*, forming part of the battery-frame. The foundation for the battery consists of heavy timbers, F, standing vertically, placed close to one another, and firmly bolted together; the side timbers, T, and the battery-posts, C, are securely fastened to the foundation, being strengthened by the iron bars, R, and stays, D. The mortars, M (fig. 208), are placed directly on the foundation of vertical mortar-blocks, and are secured by bolts, as shown in the figures. The mortar most generally employed for wet crushing is a cast-iron box, from 4 to 5 feet in length, 3 feet 6 inches in depth, and about 12 inches in width, inside measure, cast in one piece. The feed-opening, *l*, through which the ore to be crushed is introduced, is about 4 inches in width and nearly as long as the mortar. On the opposite side is the discharge aperture, furnished with a screen, *i*, through which the crushed ore is made to pass. This opening also is nearly of the same length as the mortar; the lower edge being raised from 2 to 3 inches above the tops of the dies, *d*, fig. 209. The screen is attached to a wooden frame, *j*, which is secured in grooves cast in each end of the mortar, and by two lugs, *o*, cast in front of the discharge opening; it is firmly held in its place by the wedges, *w*. Screens are sometimes placed vertically, but they are more frequently inclined, as shown in fig. 209. They may be made of brass wire-cloth, having from forty to sixty meshes to the lineal inch, but for wet crushing, sheet-iron, perforated with holes varying from $\frac{1}{24}$ to $\frac{1}{40}$ of an inch in diameter, is preferred. A piece of canvas or sheet-iron is hung loosely before the screens, to prevent splashing.

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[1] 'Mining Industry of the Fortieth Parallel, by James D. Hague, with geological contributions by Clarence King. Washington, 1870.

The mortar is furnished with dies, *d*, so fixed in the bottom as to receive the blows of the stampers, and to sustain the wear which in

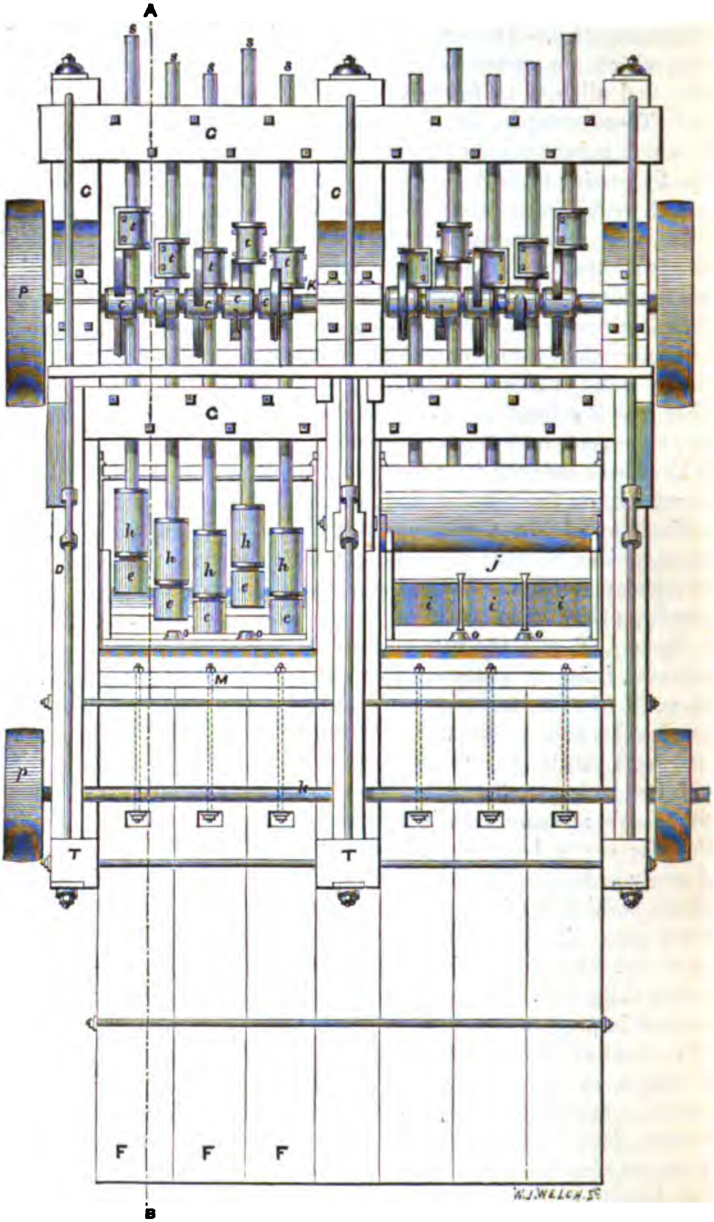

Fig. 208.—Stamping-Mill; front elevation.

their absence would be experienced by the mortar itself. Each die is a cylindrical block, about 6 inches in height, and corresponds in

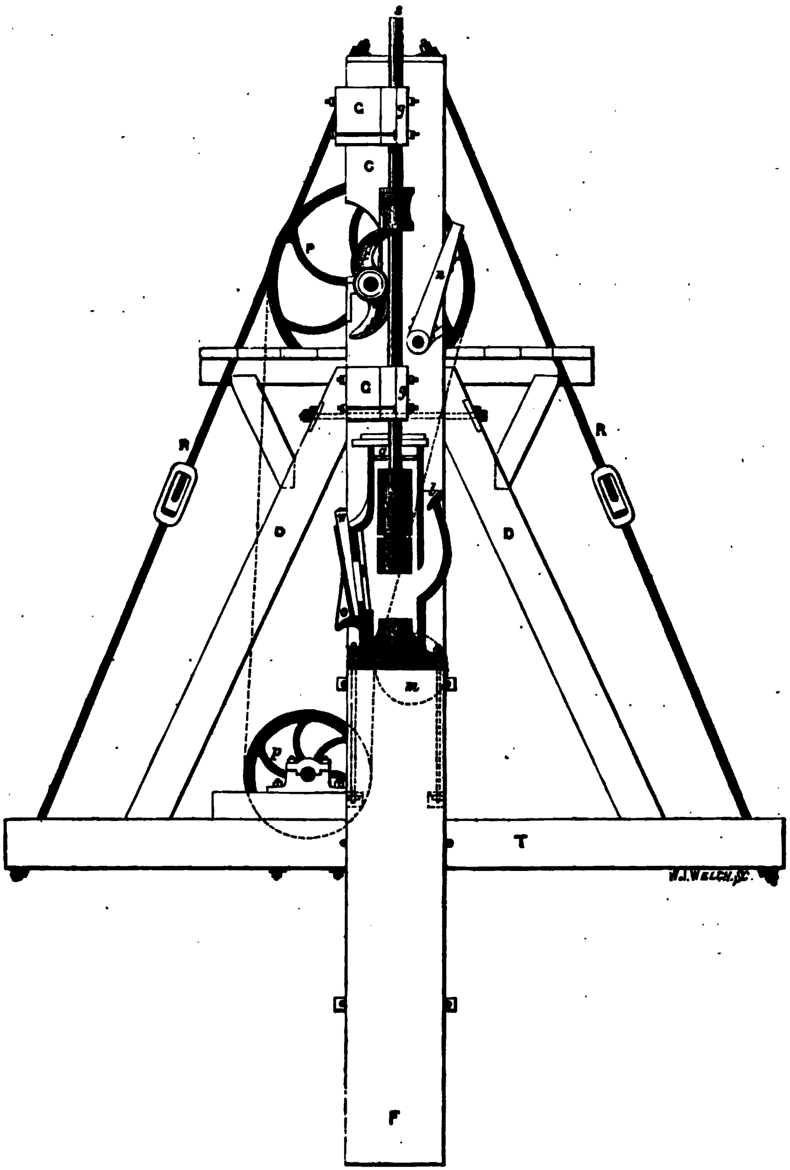

Fig. 209.—Stamping-Mill; section on A B.

diameter with the shoe of the stamper, which falls upon it. In order to keep it in its place the lower end is cast with a square flange, which fits into the bottom of the mortar and prevents it from moving. Sometimes, also, those parts of the mortar which are exposed to constant wear have a lining of cast-iron plates, which, like the dies, can be taken out and replaced when necessary. The top of the mortar is covered with planks, *q*, resting on flanges cast on the inside, which, meeting in the middle, have semicircular notches in each, so as to form apertures, through which the stems of the stampers work.

Each stamper consists of a stem or lifter, *s*, a head, *h*, attached to the lower end of the stem, and a shoe, *e*, which sustains the wear of the operation, and can be readily changed when required. It is also provided with a collar or tappet, *t*, by which the revolving cam, *c*, lifts the stamper previous to its fall. The stem is a turned bar of wrought-iron, about 3 inches in diameter and from 10 to 12 feet in length.

The stamp-head is a cylindrical piece of cast-iron, 8 inches in diameter and 15 in length, hooped at either end with wrought-iron; this hoop is shrunk into a recess, so that it does not project beyond the general surface of the cylinder. In its upper end is a slightly conical socket corresponding with the axis of the cylinder, in which the stem is secured by wooden wedges. In the lower end of the head is another socket, into which is secured, in the same way, the shank of the shoe, which is a cylinder of hard cast-iron of the same diameter as the head, and 6 inches in length; this is provided with a tapering shank 5 inches long. At the bottom of the sockets in the head for the reception both of the end of the stem and shank of the shoe, an oblong hole is cast across the diameter, for the purpose of drifting them out when necessary.

The collar or tappet, *t*, is a cylinder of cast-iron, 8 inches in length, bored out so as to closely fit the stem, to which it is tightly secured, either by a gib and keys or by some other contrivance.

The rotatory motion imparted to the stamper by the friction of the cam against the tappet is one of the advantages afforded by the use of round stems and shoes, and has not only the effect of increasing the grinding power of the machine, but also causes both the shoe and die to wear more evenly than when the stamper falls without such a circular motion. The cams, *c*, are curved arms keyed to a shaft, K (fig. 208), so placed in front of the battery that by its revolution they are successively brought in contact with the tappets on the different stampers, causing them to be raised to a height determined by the length of the cam, and fall to at the moment of release. In the silver districts of Nevada the cams are generally double-armed, as shown in the woodcuts, although single-armed cams are also employed. The form of a single-arm cam will be seen on reference to the elevation of a stamping-mill employed for treating gold quartz (fig. 222). Motion is communicated to the cam-shaft by the pulley, P, keyed upon one end of it, from which a broad belt, made of canvas and india rubber, passes over another pulley, *p*, on the driving-shaft, *k*; this belt is tightened by the pulley, *m*. The order in which the stampers are made to fall is not always the same; in a five-stamp battery, however, a common arrangement is first to let fall the middle stamper, then the end one on the right, then the second on the left, afterwards the second stamper on the right, and, finally, that on the extreme left. When it becomes necessary to hang a stamper, so that the cam may revolve without reaching the tappet, it is supported by the articulated prop or stud, *n*, of which there is one for each stamper, arranged on a small iron shaft placed across the battery and passing through the uprights, to which it is secured. Each stud is of such a length as, when placed under the tappet, to support the stamper at a height of about an inch above the highest throw of the cam. In order to bring this stud into this position the workman lays a stick, about 1½ inch in thickness, on the face of the cam as it rises towards the tappet, and holds it there while the stamper is being lifted. It is thus raised sufficiently high to allow of the support being dropped into its place, which being done, the tappet is hung above the reach of the cam. When it is desired to again set the stamper in motion the operation is repeated, the stud being withdrawn at the moment when the stick placed on the face of the cam has lifted the tappet clear of its support.

The weight of the stampers in most general use is from 600 to 700 lbs.; their usual speed is from 60 to 90 blows per minute, and their ordinary drop from 9 to 12 inches. The higher the speed of the stampers the smaller is the amount of drop given to them. A mill of this description, discharging through screens of the usual degree of fineness, will, on an average, pulverize two tons of ordinary ore from the Comstock lode, per stamper, in twenty-four hours.

The amount of water required in the battery varies with the character of the ore and with the degree of fineness to which it is to be reduced. The usual consumption in the Washoe district is from 250 to 300 cubic feet per ton of rock treated; but this includes the water used in the pans, which does not pass through the batteries. In California and throughout the gold-regions of the Pacific coast the stamping-mill employed for the reduction of auriferous quartz is similar, in all essential particulars, to that above described, and it will consequently require no further description when the treatment of gold ores has to be considered. The ore to be stamped is often supplied to the mill by a shovel through the aperture, *l*, but, as a rule, such machines are now constructed with self-feeding appliances.

Dry crushing now often replaces wet stamping in the treatment of argentiferous ores, and in such cases the fine powder which passes through the screens of the battery is usually removed by an exhausting fan and collected in dust-chambers. This method of crushing, although slower and more expensive than where water is employed, is usually preferred when the ground ore is to be chloridized previous to being amalgamated in pans. Among its other advantages dry crushing obviates the loss of finely divided, floury ore, which is always, to some extent, carried off by the water passing through the settling-pits of the wet stamping-mill.

Many of the more modern mills, both for wet and dry stamping, are, for the purpose of facilitating the egress of the ground material, furnished not only with screens in front, but also at the back, below the level of the feed-opening.

*Settling-Tanks.*—The stuff discharged from the battery is conveyed, in suspension, by the water escaping through the screens, by troughs, to settling-tanks, of which there is a series arranged in front of the stamping-mill. These tanks, which are generally constructed of planks, are 6 or 7 feet square and from 3 to 4 feet in depth; they are so disposed as to communicate with one another, near the top, in such a way that the stream carrying the crushed ore in suspension, after passing one tank, flows into the next one below, and so on from one to another. A deposit of the solid material thus takes place in each tank, until the water flowing from the last in the series escapes in a comparatively clear state. The number of tanks must be sufficient to allow of some of them being emptied, whilst others are collecting the crushed ore, and the troughs in connection with them are provided with gates, by which a certain number can be shut off from the others when required. In this way the bulk of the coarser solid particles is deposited in the tanks; but the water which escapes is still charged with slimes, consisting of ore reduced to a state of impalpable division, and which can only be collected by a process of settling extended over a considerable time. For this purpose the steam is conducted either through another series of large settling-tanks or into a large settling-pond outside the mill. The slimes thus collected form an important but variable proportion of the total amount of ore crushed, and in some instances represent as much as 10 per cent. of the material operated on; but although they afford, by assay, a considerable yield of silver, they were not for some time treated with profitable results. As soon as one of the tanks becomes filled with finely divided ore, the stream is diverted into others which have space for a further supply, and the full tank is cleaned out, the crushed ore being then subjected to grinding and amalgamation in the pan.

*Pans.*—The pans employed vary considerably in the details of construction, but all essentially consist of a round tub, the bottom of which is of cast-iron, but of which the sides are sometimes of wood. A hollow pillar is cast in the centre, within which is a vertical shaft projecting above its top, and to which motion is communicated by gearing situated below the pan bottom. To the top of this shaft is keyed a yoke or driver, by which the muller, or upper grinding-surface, is made to revolve. On the inside of the pan is fixed a false bottom of iron, cast either in sections, called "dies," or in one piece which has a diameter somewhat less than that of the pan; this has an aperture through which rises the central pillar. The false bottom furnishes the lower grinding-surface of the machine. The muller, forming the upper grinding-surface, is a circular plate of cast-iron, corresponding in size with the false bottom, and having a flat, conical, or conoidal surface, according to the shape of the pan bottom.

There are various contrivances for raising or lowering the muller, in order that it may rest its whole weight on the bottom, so as to produce the greatest grinding effect, or be maintained at any desired distance from it, when less friction or more agitation is required. Numerous devices have also been adopted for communicating a proper motion to the pulp, so that when the muller is in action the material may be constantly kept in circulation, passing between the grinding surfaces and coming in contact with the mercury with which the pan is charged. Some pans are cast with double bottoms, with an intermediate space for the admission of steam for the purpose of heating, while in others "live steam" only is employed, which is conducted directly into the pulp through an iron pipe.

The flat-bottomed pans of Varney and Wheeler, and that of Hepburn and Peterson, with a conical bottom, have been long employed with satisfactory results, although since 1860 other makers have introduced new pans, of which the characteristic features are increase of size and great simplicity of construction. Among these the large flat-bottomed pans of M'Cone, Horn, and Fountain are worthy of notice, as combining economy in first cost and a capacity which enables them to treat in the same time a much larger quantity of ore than could be worked in the pans formerly employed.

Varney's pan, which is still much used, is represented in figs. 210, 211, 212, of which the first is an elevation, the second a vertical section, and the third a view from above. The body of this amalgamator consists of a tub, A, 4 feet 4 inches in diameter and 18 inches deep, with a cover, B, in which is an opening for the introduction of the crushed ore to be ground and amalgamated. The pan is supported on suitable iron framework, and has a central conical tube, D (fig. 211), cast in one with it, extending from the bottom to a short distance above the cover, through the interior of which is a hole passing vertically through the pan, in order that the shaft, C, may work within it. On the bottom of the pan, and secured to it by the bolts, *e*, is fixed the lower muller, *a*, consisting of a circular cast-iron plate, having a round hole in the centre considerably larger than the base of the tube, D. This die or false bottom may, if desirable, be cast in sections. That portion of the aperture through the lower muller not occupied by D is filled with wood, *d*, so as to present a flat surface from the tube to the circumference. The diameter of the muller is somewhat less than that of the interior of the pan, by which means a space, *a'*, is left, to be filled with quicksilver. Above the lower muller is the upper one, *b*, of similar general form and size, having twelve shoes, *c*, the form and relative position of which will be understood by supposing a plate, of the diameter and thickness of the lower muller, attached to the under side of the upper one, to be sawn into twelve equal parts on lines drawn from the circumference of the plates to the outside of the tube, D.

The saw must also be supposed to be inclined at an angle of about forty-five degrees, thus producing grooves from the central opening to the periphery.

Each shoe is fastened to the upper muller either by a bolt, *f*, or by a wrought-iron rivet cast into the shoe and riveted into a counter-sink in the upper side of the muller, as seen in fig. 211; the bosses and recesses, *j*, keep the die firmly secured in its place.

In the lower muller are radial slots, similar to those in the upper one. These slots may be either inclined or perpendicular to the surface of the plate, and are filled with wood, set with the grain perpendicular to the plate. This wears slightly in advance of the surface of the die, thus forming cavities for the admission of pulp, by which the grinding capacity of the machine is increased.

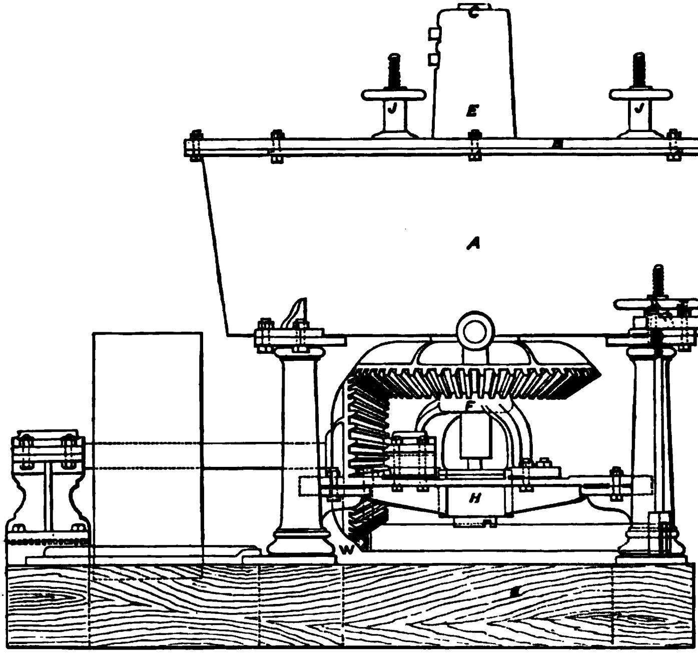

Fig. 210.—Varney's Pan; elevation.

Over and around D, but not in contact with it, is the larger conical tube, E, perpendicular to the lower face of the upper muller, and having around its lower extremity the flange, V, upon which rests the ring, *h*, forming part of the upper muller. This is connected with the muller proper by six curved arms, *i*, two pairs of which are nearer together than the others, and the space between them is filled by a projection from the periphery of the flange, V, for the purpose of carrying with it the upper muller when the flange makes a revolution. With the shaft, C, the larger casting, E, is connected by the key, *k*, and the set-screws, *l*, in the boss, G. The shaft, C, passes through a babbet-metal bearing at *m*, and through the boss, F, of the driving-wheel, in which is a key sliding vertically in the shaft. This shaft is stepped into the vertical sliding-box, H, which is itself held in the box, *o*. The step-box rests upon an iron bar, one end of which is supported by the bolt, *u*, fig. 212, and the other is connected with a screw and hand-wheel, *x*, by which it can be either raised or lowered, at the same time raising or lowering the upper muller.

Within the body of the pan are suspended three curved plates, *r*, extending from near the surface of the upper muller upwards, and stretching in length from the inner side of the pan around to a point near the outside of the larger boss, opposite to that from which they started. The lower edges of these plates are bent inwards, as shown at *s*, forming flanges, and the inner ends secured to the ring, *q*, which is of sufficient diameter to surround the tube, E, without touching it. The whole is suspended by iron rods attached to each plate, which, passing through the cover, can be adjusted by the hand-wheels, J. The outer ends of the curved plates slide vertically in grooves in the projections, *t*, fig. 212, cast upon the inner side of the pan.

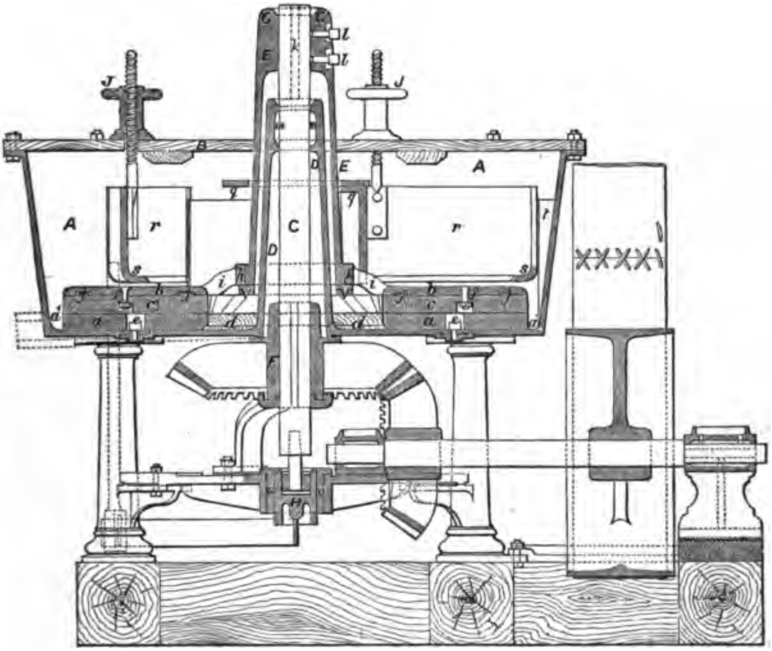

Fig. 211.—Varney's Pan ; vertical section.

The method of working the pan is as follows :—The space, *a'*, around the periphery of the lower muller, is charged with quicksilver, and the pan is nearly filled with a mixture of water and stamped ore or "pulp," in such proportions as to form an easily flowing paste. The upper muller is now made to rotate at a speed of from sixty to eighty revolutions per minute. By the centrifugal force thus developed the pulp between the mullers is made to pass through the radial channels left by the dies, as well as between the grinding-surfaces of the mullers themselves, and coming continually in contact with the mercury, with which the machine is charged, amalgamation is effected. This outward motion of the pulp prevents the mercury from coming into contact with the grinding-surfaces, by which it would be broken, and a considerable loss be the result.

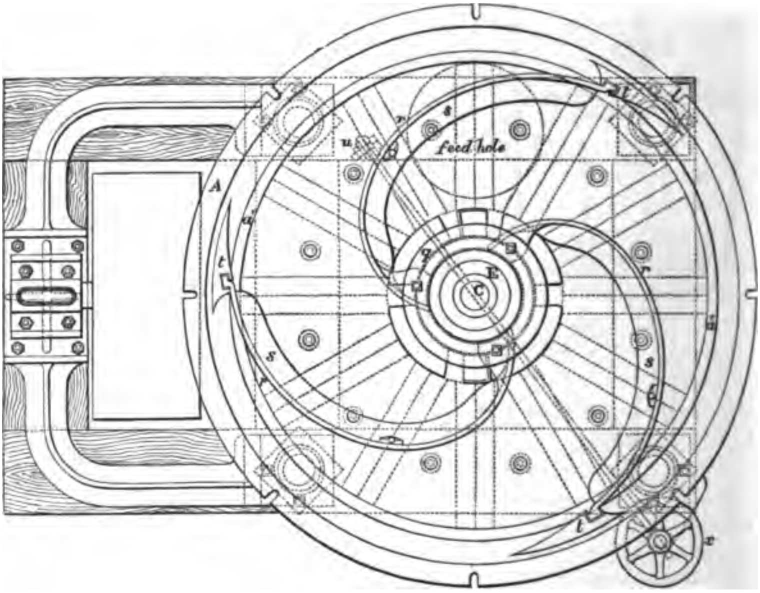

Fig. 212.—Varney's Pan; as seen from above.

The rotation of the upper muller causes the pulp in the pan to revolve with it, and this current, being met by the curved plates, *r*, is directed towards the aperture in the centre of the upper muller. The radial slots between the shoes allow currents to pass, with a sufficient velocity, to produce a partial vacuum, by which the pulp in the bottom of the pan is set in motion, causing a rapid and abundant flow downwards at the centre and upwards around the periphery of the pan. In this way the pulp is ground, and made to circulate, until the pulverization of the ore and amalgamation of the precious metals have been effected.

The Hepburn and Peterson pan has a capacity about equal to that of the Varney pan, but the form of the bottom is that of an inverted cone.

This bottom is lined with four conical dies, secured in their places in the usual way, and is never made with a steam-chamber, steam being always introduced directly into the pulp by a pipe from the boiler. In the centre is a hollow pillar, through which passes the driving-shaft. The form of the upper muller corresponds with that of the bottom, and it has, in the centre, a hollow vertical cone by which it is connected with the driver. Its under side is furnished with shoes, between which are left radial passages for the circulation of the pulp. There are also radial grooves between the shoes, cast in the muller itself, so that when they have been reduced in thickness by use there may still be a sufficient channel for the passage of pulp. A similar passage is left between the dies lining the interior of the bottom. The movable inverted cone, to which the shoes are attached, is raised or lowered by a screw and hand-wheel, the bottom of the screw resting on the top of the driving-shaft, with which the boss of the bottom pillar is connected by a sliding key. In this apparatus circulation of the pulp is effected without the use of the wings or guides employed for that purpose in some other pans. When this pan is in action the pulp, passing between the grinding-surfaces from the centre to the circumference, is made to descend towards the centre; this movement being promoted by the shape of the pan and the motion of the muller-plate. The muller is worked at the rate of from sixty to seventy revolutions per minute. The pans of M'Cone, Horn, and Fountain are flat-bottomed, and of larger dimensions than those above described; they are sometimes provided with steam-chambers beneath the bottom. The charge of the smaller pans is from 1,200 to 1,500 lbs., but those of M'Cone and Fountain, which are 5 feet in diameter and 25 inches in depth, will work charges of from 4,000 to 5,000 lbs.

In charging a pan the upper muller is raised a short distance from the bottom, water is supplied by a hose-pipe, and ore, from the settling-pits, is at the same time thrown in. The mixture is now heated, either by blowing steam into it or by a steam-chamber. In the latter case it is somewhat difficult to obtain the most desirable temperature (85° C.); on the other hand, when live steam is blown directly into the pan, attention is necessary to prevent the charge from becoming too liquid, from the accumulation of condensed water. To avoid this, in many mills, the temperature of each charge, after being first raised by the admission of live steam, is subsequently maintained by means of a steam-jacket. The muller is gradually lowered, and, in the course of about two hours, the ore will have attained the state of a fine pulp. When this has been accomplished, or, in some mills, at the beginning of the operation, mercury is introduced. This is done by pressing it through a piece of canvas, by which it becomes equally spread over the surface of the pulp in the form of minute globules. The amount thus added is generally from 60 to 70 lbs. to an ordinary charge of from 1,200 to 1,500 lbs. of ore. The muller is now raised, so as to act rather as an agitator than as a grinder, and this action is continued during two hours.

With the view of promoting amalgamation, it is usual to add to the charge, either at the time of its introduction into the pan or shortly afterwards, various materials generally known as "chemicals," and which usually consist of cupric sulphate and common salt; the quantity of each employed in different establishments varies considerably, but is usually from 1 to 3 lbs. to each charge of ore. The action of these substances, however, is but imperfectly understood, and their efficiency is open to doubt, since in many mills in which both cupric sulphate and common salt were formerly employed the use of one or the other has been discontinued, without in any way affecting the results. In other cases the employment of chemicals has been altogether abandoned, and yet, under all these varying circumstances, an equally good production of silver has been realized.

After two hours grinding, and from two to three more hours employed in amalgamation, the operation is usually regarded as complete, and the contents of the pan are run off into a separator or settler. The discharge of the pan is facilitated by the addition of water, supplied under pressure, through a flexible hose, which, at the same time, dilutes the pulp and allows it to flow readily into the separator. After having been emptied and washed out by a stream of water, the pan is supplied with a fresh charge of ore, and the operation of grinding is at once resumed.

At stated times, or whenever it is desired to ascertain the exact yield of a parcel of ore which has been under treatment, the pans, settlers, and all other apparatus containing amalgam, are thoroughly cleaned up. For this purpose the mullers must be raised, the shoes and dies removed, and all the parts scraped, in order to remove the hard amalgam adhering to the surfaces. In many cases, above one-fourth of the total amount of bullion yielded by the ore is obtained in this way.

*Separators.*—Separators or settlers, figs. 213, 214, like pans, differ to a certain extent in the minor details of construction, but are generally round tubs, of either wood or iron, with cast-iron bottoms, and resemble pans in their general features, although considerably larger, being about 10 feet in diameter. A hollow cone, C, fig. 214, is cast on the centre of the bottom, through which passes the vertical shaft, S, which is connected with gearing below the tub. To its upper extremity is attached the yoke or driver, D, which gives motion to arms, A, extending from the centre nearly to the circumference. These arms carry a number of ploughs or stirrers, P, usually made of hard wood, which rest lightly on the bottom, and when in motion communicate to the pulp the amount of agitation necessary for facilitating the separation of any mercury or amalgam with which it may be mixed. This stirring apparatus makes from twelve to fifteen revolutions per minute.

The separator is placed in front of the pans, but at a lower level, so that the latter may be conveniently discharged into it. In some establishments two pans are discharged at the same time into one settler, in which case the operation occupies the same length of time as the grinding and amalgamation of a charge in the pan, or from four to five hours. In other mills only about two hours are allowed for settling, and the two pans connected with each settler are discharged into it alternately.

The water employed in discharging the pan considerably dilutes the consistency of the pulp, and this dilution is often further increased by the addition of fresh quantities during the progress of the operation. The degree of fluidity of the pulp, and the speed of the stirrers, materially affect the results, since if the paste be too thick the amalgam and quicksilver will remain in suspension, and if, on the contrary, it be too thin, sand will settle with them on the bottom of the vessel. It is evident that a too rapid or a too sluggish motion of the revolving arms would produce similar effects. The degree of dilution yielding the most satisfactory results with a given speed of the agitator can only be determined by experience. A discharge-hole, near the top of the tub, allows the lighter portions of the pulp to flow off, and at successive intervals the point of discharge is lowered by withdrawing a lower plug from a series of holes, *h*, in the side of the settler. In this way the whole of the materials, with the exception of amalgam and quicksilver, are finally removed. The two latter are subsequently collected by the aid of various devices. There is for this purpose generally a groove in the bottom of

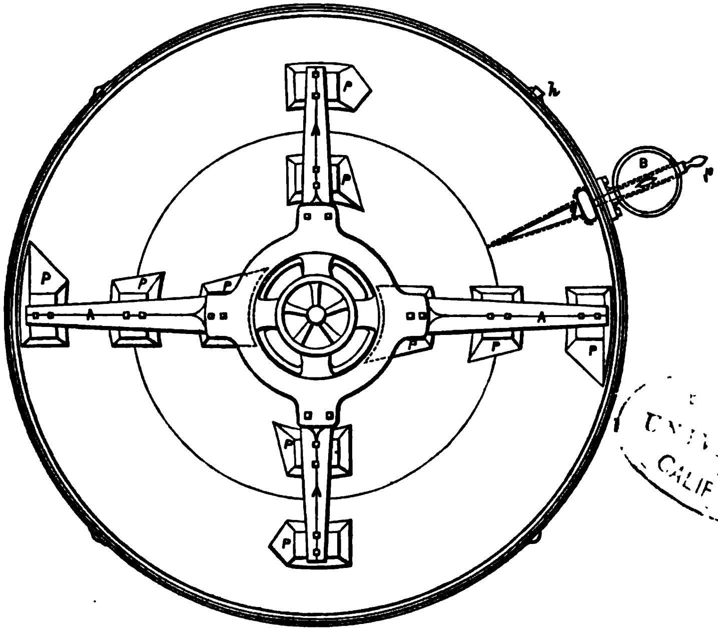

Fig. 213.—Separator; as seen from above.

the separator, leading to the bowl, B, from which the fluid amalgam may be dipped; or it may be drawn off by removing the plug, *p*, from the end of the outlet-pipe. The quicksilver, charged with amalgam, is cleaned by repeated washings with water, and by carefully removing from its surface any particles of sand, pyrites, &c., that may adhere to it. In some mills the cleaning of the quicksilver and amalgam is effected in a small iron pan, resembling a settler in its construction, in which it is washed by slow agitation with plenty of clean water. When sufficiently cleansed, the amalgam is separated from the redundant mercury by straining through a canvas bag, of the form and dimensions of that employed in Mexico for a similar purpose, which has been described when treating of patio amalgamation.

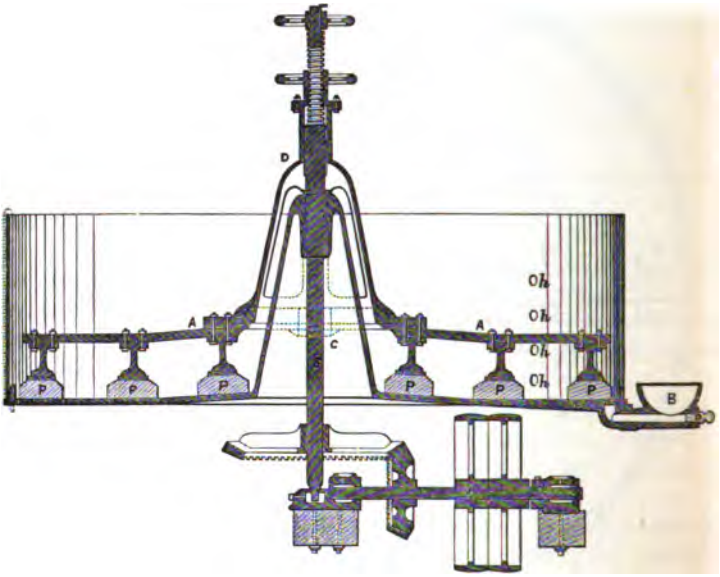

Fig. 214.—Separator; vertical section.

*Agitator.*—After leaving the separators the pulp is passed into wooden tubs, varying from 6 to 12 feet in diameter, and from 2 to 6 feet in depth, in which are collected small portions of mercury and amalgam, as well as heavy particles of undecomposed ore, which have been carried off in the pulp discharged from the separators. A simple stirring apparatus, somewhat resembling that of the separator, keeps the material in a state of gentle agitation; the revolving shaft carries four arms, and makes from ten to fifteen revolutions per minute. Some establishments have several agitators, in many cases only one is used, while in others they are entirely dispensed with. The stuff that accumulates on the bottom is shovelled out, at intervals of three or four days, and is again worked over in the pans. Beyond the agitators are blanket-sluices and various other contrivances for concentrating and collecting the more valuable portion of the tailings.

*Retorting and Melting.*—The amalgam having been strained in bags, and pressed, in order to expel as much as possible of the fluid quicksilver, is subjected to a process of distillation, by which the remaining portion is separated from the gold and silver. The cast-iron retort employed for this purpose is from 2½ to 3 feet in length, and from 9 to 12 inches in diameter, the casting being about 1½ inch in thickness. This rests either on two heavy cast-iron bearers, the ends of which are built into the brick-work, or on an arch of fire-bricks, and is placed beneath another arch, from the crown of which the products of combustion are carried off to a chimney, through rectangular apertures in communication with a flue. The open end is fitted with a cover like that of a coal-gas retort, and from the other end an iron tube carries off the volatilized mercury. This is screwed to a downcast pipe, and is so arranged that, by means of screw stoppers, every facility is afforded for cleaning the pipes. The vertical tube is enclosed within another, so as to form a Liebig condenser, through which a current of cold water is constantly passed, the heated water escaping at the top.

The downcast pipe opens into a small chamber without a bottom, sufficiently immersed in a vessel of water to keep it air-tight, but only to such a depth as to prevent the occurrence of accidents from the passage of water up into the heated retort.

This retort is provided with cast-iron semi-cylindrical trays, which are easily slid into their places, and are generally divided into two parts by a transverse partition. In some cases the amalgam is introduced directly into the retort, the use of trays being dispensed with.

Before the amalgam is placed in the retort or trays, the interior is coated with a thin wash of clay or of milk of lime, or a lining of paper may be employed instead. By this precaution the retorted amalgam is prevented from adhering to the iron, and much trouble avoided. The amalgam, having been placed in the retort, the cover is luted either with clay or with a mixture of clay and wood ashes, and is fastened in its place by a screw-clamp, or otherwise. A fire is now lighted, and the heat slowly and gradually raised, until the retort assumes a bright cherry-red colour, and is so maintained until mercury ceases to distil over. This usually occurs at the expiration of eight hours, and the charge of amalgam operated on, at one time, may vary from 800 to 1,200 lbs. The retort is now allowed to cool gradually, and, when sufficiently cold, the crude bullion is withdrawn. This amounts to about one-sixth the weight of the original charge.

A form of retort frequently employed in the neighbourhood of Virginia City is represented, in longitudinal section, in fig. 215. The ash-pit, A, is beneath the fire-place, B, which communicates, by the flues, *a*, with the chamber, *b*, enclosing the retort, C, from which the products of combustion are conveyed by the flues, *c*, through the cavity, *d*, to the chimney. These flues have regulating dampers so as to keep the retort uniformly heated throughout its length. The tube, D, conducts the mercurial vapours to the vertical pipe, E, where they are condensed by the current of cold water flowing through F. The condensed mercury collects in the reservoir, G, from which it is drawn off through an iron pipe. Any mercurial vapours that may escape through leakage, or the removal of the door, are collected by the hood, *e*, and conveyed to the chimney. The arrangement for securing the door, *g*, is similar to that employed for gas-retorts. The trays, *h*, are used for holding amalgam, and the iron braces for binding the brickwork are indicated by the letter *f*.

The retorted amalgam is broken up, melted, and cast into ingots, the fusion being most commonly effected, with the addition of a little borax, in an ordinary plumbago crucible. The loss in weight experienced in melting retorted amalgam is between 2 and 3 per cent. The ingots thus obtained are chipped and assayed in the usual way, and commonly contain, in 1,000 parts, 24 parts of gold and 840 of silver, the remaining 136 thousandths consisting chiefly of copper.

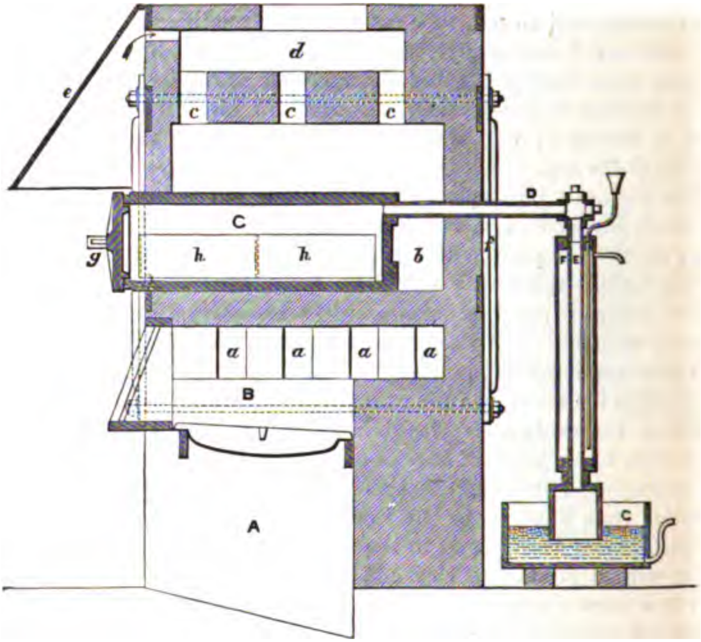

Fig. 215.—Retort and Setting; longitudinal section.

*Tailings.*[1]—The pulp, after issuing from the settlers, in which it has been, as far as possible, separated from amalgam and mercury, is variously treated in different mills. In some the whole mass is passed through agitators, for the purpose of collecting a portion of the amalgam, mercury, and undecomposed sulphides carried off from the separators. In others concentrators of various kinds are employed with a similar object, by the use of which a certain amount of undecomposed sulphides is obtained in a concentrated form. In cases where there is a sufficient supply of water, and the inclination of the surface admits of such an arrangement, blanket-sluices are laid down, over which the stream of tailings is allowed to flow; the heavier and more valuable particles being arrested by the blankets. Dams are also constructed at convenient points for the accumulation of tailings, which, after exposure to the weather for several months, may be again worked at a profit.

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[1] The term "tailings" is applied to the pulpy and sandy residues which escape from the separator or agitator after the treatment of ores by pan amalgamation.

The ordinary yield obtained by pan amalgamation varies between 65 and 70 per cent. of the assay value of the ore, and this, by the subsequent treatment of slimes and tailings, is sometimes increased to 85 or 90 per cent. In the Washoe district the cost of treatment, where water power is employed, is from $5 to $5.50, and in steam-mills from $6 to $6.75 per ton. The tailings, &c., collected in the various reservoirs established for that purpose, contain, on an average, by assay, gold and silver of the value of $15 per ton, from which $9.75 are extracted, by re-treatment, at an expense, in steam-mills, of about $5.50 per ton.

*Chemical Reactions of the Washoe Process.*—Mr. James D. Hague, who has carefully investigated this subject, has arrived at the following conclusions:[1]—

That the ores treated chiefly consist of native gold, native silver, and sulphides of silver, associated with varying proportions of blende and galena.

The action of sodium chloride and cupric sulphate produces in the pan cupric chloride.

The presence of metallic iron causes the formation of cuprous chloride.

Both cupric and cuprous chlorides assist in the reduction of the ore, by the chloridizing of silver sulphide, and by decomposing blende and galena.

Cupric sulphate augments the amalgamating energy of mercury by the formation of small quantities of copper amalgam, and also tends to remove lead from the quicksilver.

Notwithstanding the action of these reagents, as above indicated, the quantities usually added in the Washoe mills are too small to produce any very beneficial results.

Mercury and iron, under the influence of heat and friction, are the

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By "slimes" is generally understood those portions of the ore which have been reduced in the battery to such an impalpable state as to be carried through the settling tanks in suspension in water. The more pulpy portions of tailings are sometimes called "pan-slimes."

[1] 'Mining Industry of the Fortieth Parallel,' p. 293.

chief agents in the extraction of the precious metals by the Washoe process.

It is an essential condition that the mercury be kept perfectly bright and pure, in order to effect direct contact of that metal with silver sulphide and metallic iron.

In the Washoe process the consumption of mercury is chiefly mechanical, the loss through chemical action being comparatively small.

*General Arrangement of Reduction Works.*—The batteries are commonly arranged in one straight line, behind which, on the feed side, is the breaking-floor, where the ore is reduced to a suitable size for the stamping-mill, either by a mechanical stone-breaker or by hammers. When the slope of the ground permits such an arrangement, large bins are frequently constructed at a higher level behind the breaker, into which are tipped the contents of the waggons bringing ore out of the mines. The stamping-mills discharge the crushed ore into troughs, which convey it to settling-tanks standing in front of the batteries, and a platform is provided for the reception of the ore shovelled out of the tanks.

In the majority of cases the pans are arranged in a straight line, parallel to the batteries, while the separators stand in another line parallel to the pans, and on a sufficiently lower level to admit of the contents of the pans being tapped directly into them. Below the separators are the agitators, or other contrivances for preventing the escape of ore and amalgam. Power is communicated from a steam-engine or water-wheel, either by gearing or by belting, to a shaft in front of, and parallel with the batteries. On this shaft are pulleys opposite to those on the several cam-shafts, to which motion is communicated by suitable belting. The same shaft imparts motion, through a counter-shaft and belting, to the stone-breaker and pans. For the purpose of working the latter a line of shafting is arranged under them, from which the various separators and agitators are also driven by belting and pulleys.

The power required for each stamper of ordinary weight is about 1½-horse, whilst that necessary for each pan varies from 3- to 6-horse power, according to its size and construction. The expenditure per ton of ore stamped, ground, and amalgamated varies according to the size of the mill and the degree of perfection of its arrangements, but may be taken on an average at 2-horse power.

In Arizona and Sonora some mills are arranged on Boss's system, which dispenses with the operation of tank shovelling. The pans are arranged in sets of eight, each one being connected with that next below it by an overflow pipe placed 7 inches below the top edge of the tub. The coarse sand in the battery pulp is separated by a current sizing apparatus, and passed through Nos. 1 and 2 pans, where it is ground fine without mercury, and then to No. 3, which also receives the fine slimes from the battery. Mercury is first added in No. 3, and is also used in the remaining five pans, Nos. 4 to 8, but the grinding is only continued as far as No. 5, the last three pans being worked with the mullers lifted. The ore passes through the whole series of pans in four hours. About 200 lbs. of fresh mercury are added to each pan at intervals of two hours; this is done by hydraulic pressure, the mercury being contained in a closed tank, which is connected by a system of pipes with the pans on one side and with the mud drums at the bottom of the steam boilers on the other.

In Kröncke's process, used at Antofagasta and Copiapo in Chili, the ore, which need not be previously roasted, but must be finely pulverized and dried, is treated with a freshly prepared solution of cuprous chloride in common salt, mercury, and zinc- or lead-amalgam, in barrels like those of the Freiberg process, but much larger, holding from 3 to 4 tons instead of 10 cwts. Only sufficient water is used to bring the charge to the consistence of a moderately stiff paste, and the cuprous chloride solution is allowed to act upon the ore for about half an hour before the mercury and zinc are added. The operation is finished in about six hours, when the amalgam is collected and cleaned in the usual way. The use of zinc and lead prevent loss of mercury by decomposing calomel with the production of the chlorides of the less valuable metal, and they are also supposed to expedite the amalgamation by electro-chemical action. Lead is only used when the ores contain native chloride and bromide of silver.

## THE STETEFELDT FURNACE.

In many localities the most expensive item in the cost of working first-class ores by barrel amalgamation was roasting or chloridizing, which alone was in the Washoe district generally estimated at about $15 per ton. Some years since a furnace was invented by Mr. Stetefeldt, of Austin, which at one time promised to effect a considerable saving in the expense of this operation. Its action consists in allowing very finely pulverized ore, mixed with common salt, to fall against a current of heated air rising through a shaft, by which the particles of the metallic sulphides are transformed into oxides, whilst sulphurous anhydride is evolved. By the action of this and watery vapour on common salt, hydrochloric acid is generated, and by the reaction of this acid on the oxides, metallic chlorides are obtained.

The chemical action of this apparatus is very similar to that of the reverberatory furnace, but, as the ore is made to fall in a shower of finely divided particles, it is more thoroughly exposed during its descent to oxidizing and chloridizing influences. In consequence of this its action is stated to be very rapid and complete, while the expenditure of labour is said to be small.

This furnace consists of a shaft, 20 feet in height and from 3 to 4 feet square at the base. At two opposite sides are fire-places, from which short flues communicate with the main shaft. At the top is the feeding apparatus, which supplies a continuous shower of the ore in a state of extreme division. At a short distance below the top of the shaft is a flue, through which the gases escape, and by which they are conducted to a series of chambers, where any portions of the ore which may have been carried over by the draught are deposited. An auxiliary fire-place, in communication with the flue, serves the double purpose of keeping up the temperature and of extending the region of chemical action.

A discharging-door is left at the bottom of the main shaft, whence the principal portion of the ore is withdrawn. Similar doors are arranged at convenient points along the main flue, and communicate with the several chambers. The chimney for the final escape of the gases at the end of the dust-chambers is about 40 feet in height.

The ore is mixed with salt on a drying floor, and then crushed by dry stamping. It is afterwards raised by an elevator to the hopper of the feeding apparatus at the top of the furnace, whence it is supplied continuously to the chloridizing column. The temperature of this is maintained as uniform as possible, the heat employed being sufficient to keep the ore, which accumulates at the bottom, constantly red hot. Mr. Stetefeldt states that the results of the actual working experience of one of his furnaces, which he erected at Reno, go to show that it gets through a larger amount of work with a smaller expense for labour, fuel, and salt, than any apparatus previously employed for the purpose. It has been further stated that 90 per cent. of the silver present in the ore is converted into chloride. One of these furnaces, worked by eight men, is said to have accomplished the chloridizing of as much ore as ten reverberatory furnaces requiring the labour of thirty-six men. The fuel consumed in the Reno furnace averaged about two cords of wood in twenty-four hours, and treated an amount of ore per diem which, in ordinary calciners, would require the consumption of ten cords. From 3 to 6 per cent. of salt was required, according to the richness of the ore; while in the reverberatory furnace at least twice that quantity would be necessary. It is also maintained that the bullion produced from ores roasted in this furnace contains less impurity than that from those treated in the ordinary way, and consequently that it is well adapted for working ores containing a large amount of "base metal." The expense of roasting 1 ton of ore with salt in the Reno furnace was, in 1870, given as between $6 and $7; but it was expected that this cost would be materially reduced by the application of certain projected improvements.

The woodcuts, figs. 216, 217, afford a tolerably correct idea of the construction of the furnace at Reno. *a*, Shaft through which the ore falls; *b*, top of shaft, on which the feeding apparatus is arranged; *c*, damper, inserted when the screens of the feeding machinery are exchanged; *d*, door through which the roasted ore is discharged upon the cooling floor; *e*, fire-places; *f*, flue through which the gases escape; *g*, triangular flue-bridges of cast-iron; *h*, cast-iron plates, forming the bottom of the flue, and which allow the dust, which settles in this part of the apparatus, to fall into the chamber, *i*; *k*, discharging door; *l*, fire-place, which heats the lower portion of the flue, *f*; *m*, flue connected with the dust-chamber, *o*; *n*, discharging doors.

The principal dust-chamber is 24 feet long, 8 feet wide, and 10 feet high; from this the gases pass under the floor of a kiln, on which the ore and salt are dried, 39 feet in length and 7 in width. A flue 3 feet 4 inches wide, 4 feet 6 inches high, and about 180 feet in length, leads from the drying-kiln to an iron chimney, 2 feet 6 inches in diameter, situated on the hill-side. The top of this chimney rises 40 feet above that of the furnace. The fire-places and arches are built of fire-bricks, but the other parts of the apparatus with common bricks. The walls are built double, with spaces between them, and the furnace is bound with iron rails and $\frac{7}{8}$-inch rods.

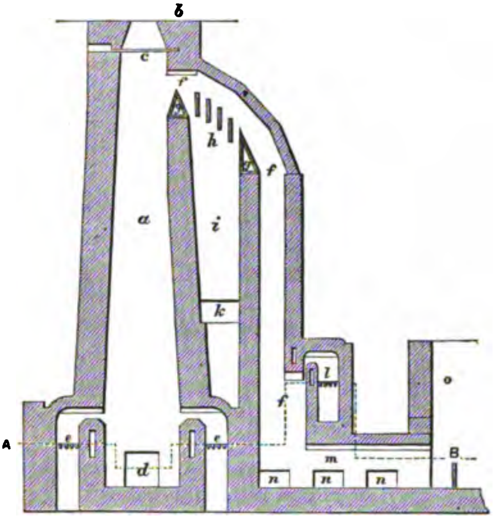

Fig. 216.—Stetefeldt Furnace; vertical section.

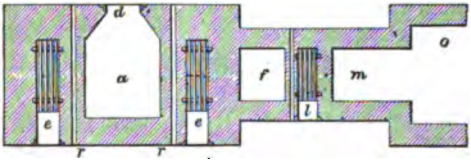

Fig. 217.—Stetefeldt Furnace; section on A B.

At first much difficulty was experienced in providing suitable feeding apparatus. Gerstenhöfer's feeder, consisting of fluted rollers, which force the ore through slits on the top of the furnace, was not found to answer, as it caused the ore to fall in lumps, which arrived at the bottom of the shaft in an almost unaltered state. This is caused by the tendency possessed by the particles of all finely pulverized minerals to adhere to one another if a slightly compressed mass be allowed to fall through the air.

After various trials the apparatus for this purpose was arranged as follows:—

A hollow cast-iron frame, kept cool by a small stream of water, rests on top of the furnace. In this frame is a cast-iron grating, covered by a screen of finely punched sheet-iron, similar to those employed for wet crushing. Immediately above the punched screen is another, made of coarse wire-cloth, fastened to a movable frame. This is provided with flanges resting on adjustable rollers on the outside of the hopper, and receives a reciprocating motion from a crank. The throw of this is 1⅝ inch, and in order to avoid the motion of a stratum of pulverized ore with the coarse sieve, a number of thin iron blades are so arranged across the hopper that their lower edges almost touch the upper surface of this sieve. These blades keep the finely divided ore from being displaced when the crank is in motion, while the meshes of the iron screen cut through it and cause its particles to fall through the apertures of the punched screen beneath. The number of revolutions of the crank-shaft varies from thirty to seventy per minute, and the ore is thus regularly and continuously introduced into the furnace.

In the newer forms of this furnace the shaft and dust flue are raised above the ground upon piers and terminate below in hoppers, which discharge the roasted ore into waggons placed below. They are made with shafts, 30 to 40 feet high, and roast from 20 to 40 tons of ore in twenty-four hours.

## Amalgamation of Refractory Ores in Montana.

In the territory of Montana large quantities of silver ore containing a little gold are raised. These ores, consisting mainly of sulphides of silver, antimony, copper, lead, and zinc, are locally known as "base ores," the assay of which varies from 40 to 50 ounces of silver, and from 3½ to 10 dwts. of gold per ton of 2,000 lbs. The mill arrangements for the treatment of these ores usually include grizzlies or screens, stone-breakers, drying cylinders and flues, automatic stamp-feeders, dry stamping-mills, chloridizing furnaces, amalgamating pans, settlers, and bullion retorts. The drying cylinders under the grizzlies and stone-breakers, usually of wrought-iron 18 feet long, 42 inches tapered to 56 inches inside diameter, make about eight revolutions per minute. The "Howell and White" chloridizing furnace, 28 feet long and 60 inches inside diameter, slowly revolving between a fire-place and dust-chambers, is but a slight modification in form of the Hocking and Oxland calciner, used in the tin mines of Cornwall, described at p. 476.

The mortar-boxes of the stamps are fitted with front and back screens, each enclosed in wooden chambers. At the bottom of the chambers is an Archimedean screw for delivering the ore to an elevator fixed at the end of the battery. The ore from the drying cylinder is weighed to the stamps, and the salt is then added, the quantity varying, with the baseness of the ore, from 10 to 14 per cent. by weight. Both ore and salt are reduced in the mortar to a dry dust. This saline dust is elevated so as to be passed direct to the chloridizing cylinder or furnace. From this furnace it is discharged into storage boxes, from which charges are drawn for the amalgamating pans.

A charge is amalgamated and discharged into the settler in from six to eight hours. Sometimes the Hasenclever furnace is employed for drying the ore previous to its being mixed with salt. When the Stetefeldt furnace is used for chloridizing purposes, the ore and salt drop from the top to the bottom of the shaft, about 60 feet. In the following table will be found the approximate weight and cost of chloridizing ore in four distinct mills, at the prices stated for labour, salt, and fuel, as well as the assay proportions of silver chloridized and amalgamated, together with the assay proportion of gold obtained. The ore chloridized in the Stetefeldt furnace was very base, requiring 14 per cent. of salt.

| Mills. | A. | B. | C. | D. |
|---|---|---|---|---|
| Character of furnace used | Stetefeldt. | Howell & White. | Howell & White. | Howell & White. |
| No. of tons of 2,000 lbs. each chloridized per twenty-four hours | 25 | 30 | 28½ | 20 |
| | £ s. d. | £ s. d. | £ s. d. | £ s. d. |
| Labour, at 14s. per day | 6 6 0 | 4 8 0 | 4 8 0 | 1 14 0 |
| Salt, at £8 per 2,000 lbs. | 28 0 0 | 24 0 0 | 18 4 9 | 16 0 0 |
| Wood fuel, at 26s. per cord | 5 10 6 | 4 17 6 | 4 11 0 | 3 18 0 |
| Power | 0 10 0 | 0 12 0 | 1 6 0 | 1 6 0 |
| Oil and light | ... | 0 3 0 | ... | ... |
| Wear and tear | ... | 1 0 0 | 1 0 0 | 1 0 0 |
| Interest | 1 12 0 | 1 2 0 | 1 2 0 | 1 2 0 |
| | 41 18 6 | 36 2 6 | 30 11 9 | 25 0 0 |

| | A. | B. | C. | D. |
|---|---|---|---|---|
| Average cost per ton of 2,000 lbs., | £1 13 6 | £1 4 0 | £1 1 6 | £1 5 0 |
| | Per cent. | | Per cent. | Per cent. |
| Silver chloridized | 83 | ... | $82\frac{3}{10}$ | 84 |
| Silver amalgamated | $86\frac{8}{10}$ | ... | ... | ... |
| Gold " | 60 | ... | $56\frac{7}{10}$ | ... |

## PROCESSES FOR EXTRACTING SILVER BY THE WET WAY.

The processes by which silver is extracted from ores and metallurgical products by the various wet methods are all comparatively modern, and belong to that recent period during which the practical metallurgist has availed himself of the assistance to be derived from chemical research. These methods have now, to a great extent, supplanted the older processes of liquation and amalgamation, and may often be advantageously employed for the treatment of argentiferous materials, particularly when the amount of copper is large, and that of lead comparatively small. Under certain conditions the methods of silver extraction by the wet way possess advantages over smelting and amalgamation; but in the case of some of them, in order to obtain satisfactory results it is necessary that the material operated on should not contain any considerable amount of either lead, zinc, antimony, or arsenic. Ores containing these metals in large quantities are not adapted for treatment by any of the wet processes, and the fact of their being frequently associated with silver ores has tended to restrict the application of such processes.

AUGUSTIN'S PROCESS.—When ordinary argentiferous ores, or sulphuretted metallurgical products containing silver, are roasted with common salt, chloride of silver is formed, which is soluble in a hot and concentrated brine. From this solution the silver may be precipitated by metallic copper, which can in its turn be thrown down by iron. The residual liquors, until by repeated use they contain too much sodium sulphate, may be employed for dissolving fresh quantities of silver chloride. The solubility of silver chloride in a solution of common salt is a fact long known, but was first taken practical advantage of (1849) by Augustin, one of the officers of the Mansfeld Mining Company, as the foundation of a process for the extraction of silver from its ores. This process is less applicable to the direct treatment of ores than to mattes, since raw ores frequently contain substances which interfere with the complete conversion of the silver into chloride.

Copper mattes, yielding from 50 to 70 per cent. of copper, but free from metallic granules, and containing neither lead, zinc, antimony, nor arsenic, afford the best results by Augustin's process.

The presence of rich copper sulphides is favourable to the production of residues poor in silver, but a mixture of metallic copper, in the form of shot, necessarily results in a loss of silver. It is often found advantageous to submit copper mattes to concentration before subjecting them to treatment by this process. When lead is present, it may be transformed into chloride and removed by hot water, previous to the treatment of the roasted mattes by brine.

The desilverization of copper matte is effected by the following series of manipulations:—

*First Roasting.*—The matte is first ground and sifted, and then roasted at a low red-heat on the upper bed of a double reverberatory furnace. This operation is completed in about five hours, the ordinary charge being about 4 cwts. The ore is then transferred to the lower bed, where it is roasted during two hours at a moderate temperature. The heat is then raised, and the roasting continued during three additional hours. By this treatment the silver contained in the matte will, for the most part, be transformed into sulphate, while the corresponding iron and copper salts will be decomposed into oxides mixed with basic sulphates. A sample drawn from the roasted charge should, when treated with hot water, afford a solution of a faintly blue colour, which by the addition of a drop of solution of common salt should give a precipitate of silver chloride.

*Roasting with Salt.*—As soon as a sample taken from the furnace affords the results above described, the charge is withdrawn, and after being allowed to cool, is ground between millstones. The ground ore is then passed through a bolting-sieve, and placed in the same furnace in which it was previously treated, where it is mixed with from 3 to 5 per cent. of common salt. It is now roasted at a low temperature, by which chloride of sodium is decomposed by the sulphuric acid of the sulphates. Chlorine unites with silver, and nearly the whole of that metal is transformed into the state of chloride. This second roasting occupies from two to three hours, and the mixture, after being withdrawn from the furnace and allowed to cool, is taken to the lixiviating house.

*Lixiviation and Precipitation.*—A lofty shed should be used for the lixiviation of the chloridized ores. At Freiberg, where Augustin's process was employed from the year 1849 to 1862, when it was abandoned in favour of a process for the treatment of roasted mattes by sulphuric acid, the operations were conducted with a plant of which fig. 218 represents an elevation.

The lixiviating tubs, *a*, arranged in a straight line on a floor considerably above the ground-level, were each provided with a false bottom supporting a filter. On the bottom of the tub was laid a wooden cross, upon which rested a disc made of planks perforated with large holes. This was covered by a uniform layer of twigs, and a linen cloth was stretched over them and made tight against the sides of the vessel by a wooden hoop. These tubs, provided with wheels, were each charged with about 8 cwts. of roasted matte, and could be transported by the bogie, *b*, running on the tramway, *c*, and were finally arranged in their respective places by cross-rails on the platform, *d*. Hot brine was conducted to the several tubs from the reservoir, *e*, supplied from the larger tank, *f*, through the trough, *g*. The solution of salt was heated to the necessary temperature by steam-pipes. The heated brine, on coming in contact with silver chloride, dissolved it, and flowed off through the filters into a trough, by which it was conducted into a tank above the level of a series of tubs. From this reservoir the fluid was conducted into four tubs, not shown in the woodcut, in which the silver was precipitated by cement-copper. The copper used for this purpose was placed on filters similar to those on the bottom of the lixiviating tubs. From these tubs the liquors flowed into three tubs, *h*, charged in the same way as the upper ones with precipitated copper, where the last traces of silver were thrown down. The cupriferous liquors now falling successively into the series of tubs, *i* and *k*, charged with metallic iron, deposited the greater portion of their copper, and were finally conducted into a tank, *l*, where any traces of copper still retained in solution were precipitated by an additional supply of iron scrap. The brine, thus freed from silver and copper, was pumped back again into the reservoir, *f*, to be re-heated and again used.

When the tubs, *a*, had been exhausted of silver, they were taken on the bogie, *b*, and placed on a line of rails at right angles to the tramway, *c*, where they were washed, first with liquors resulting from previous washings, and finally with water. The washing waters, when sufficiently concentrated by repeated use, were treated as silver solutions. After the third lixiviation, for which pure water was employed, the tub was taken to the tipper, *m*, where it was turned over and its contents discharged into a drainer, *n*.

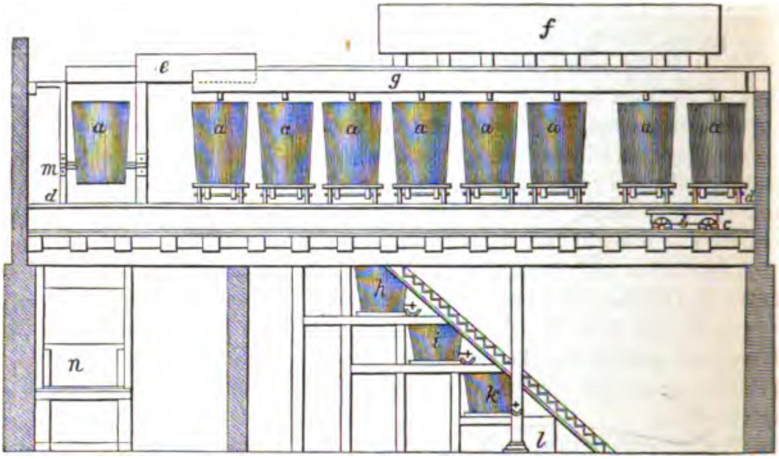

Fig. 218.—Augustin's Process; side elevation of plant.

The process of lixiviation may be divided into two periods: the first, during which the ore is treated with concentrated hot solutions of common salt, occupying about twenty hours. The first period was considered as terminated when a piece of bright copper held in the escaping liquor was no longer whitened by a deposit of silver, and the tubs were then removed upon the bogie, *b*, for the purpose of receiving the second washing; first with weak liquors, and afterwards with water, as above described.

The products obtained were—Firstly: residues in tubs, containing from 40 to 65 per cent. of copper, with more or less considerable traces of silver. When found to contain more than 0·03 per cent. of silver they were put aside to be again operated on. When affording less than that quantity they were passed to the smelting department for the production of copper.

Secondly: cement silver, in a finely divided state, which, after being washed with dilute hydrochloric acid, and subsequently with water, was pressed into balls, thoroughly dried, and taken to the refinery.

Thirdly: cement copper, employed for the precipitation of silver during succeeding operations.

Fourthly: liquors freed from silver and copper, from which a portion of the iron precipitates as a basic salt; these liquors can be employed in lieu of fresh brine, but require to be occasionally freed from sodium sulphate by crystallization.

The expense of treating copper matte by this process necessarily varies in accordance with the cost of salt, fuel and labour, in the locality in which the works may be situated. The loss of silver is from 8 to 12 per cent. Speiss yields its silver to this process with more difficulty than copper matte.

Ziervogel's Process.—Augustin's process for the extraction of silver by hot brine, after being in operation for some years at Mansfeld, was superseded in the year 1857 by a simpler and cheaper method introduced by Hüttenmeister Ziervogel, which has been found to be peculiarly well fitted for the treatment of the very pure matter produced there. This is also known as *Wasserlaugerei*, or water lixiviation, water being used as a solvent instead of chloride solutions.

This method is founded on the circumstance, that when a mixture of copper and iron sulphides, containing silver, is roasted in a state of fine division, in a reverberatory furnace, ferrous sulphate is first formed. This, by further roasting, becomes ferric sulphate, which is finally decomposed into ferric oxide. At this period sulphide of copper is transformed into cupric sulphate, and on the temperature being further increased, cupric oxide is produced and sulphuric acid expelled. Finally, silver sulphide is converted into silver sulphate, a salt readily dissolved in water, while nearly all the other ingredients of the roasted matte are insoluble in that menstruum. If the roasted material be now lixiviated with hot water, the silver will be obtained in a solution, from which it can be precipitated in the metallic form.

Ziervogel's process, although well adapted for the desilverization of the pure copper mattes of Mansfeld, is not generally applicable to the treatment of silver ores contaminated by arsenic, antimony, lead, or zinc.

The quantity of silver in the refined copper produced at Mansfeld, from residues partially resulting from amalgamation, and partly from treatment with hot brine, was (1846–1849) from 0·0388 to 0·0631 per cent.; while that resulting from Ziervogel's process contains now only 0·029 per cent.

The copper matte operated on at Mansfeld is first concentrated, and afterwards reduced to the state of impalpable powder. Its average composition is nearly as follows:—

| | |
|---|---|
| Cu | 74·51 |
| Ag | 0·43 |
| Pb | 0·55 |
| Fe | 2·30 |
| Ni | 0·30 |
| Co | 0·19 |
| Zn | 0·51 |
| Mn | 0·01 |
| S | 20·70 |
| | 100·00 |

The matte, after being ground, is bolted through cylindrical sieves, having from 1,400 apertures per square inch, and all particles too coarse to pass through the meshes, escape at the lower end of the cylinder, and are returned to be re-ground.

*Roasting.*—This operation is conducted in a furnace provided with two beds, each 10 feet in length and 8 feet in width, placed over one another. The upper bed is heated from below by the flame and gases passing over that beneath it, whilst from above it receives its heat from the same gases, which are conducted over its arch in high zigzag flues, answering the purpose of condensing chambers. Above these flues are cast-iron plates, forming a floor on which the discharged residues are dried previous to being smelted for copper. The gases are finally conducted into a high chimney, which is in communication with several similar furnaces. In order to regulate the admission of air beneath the grate the ash-pit is closed, but is in connection with an arched channel running below the furnaces, and communicating with the atmosphere. The amount of air admitted into the ash-pit through this passage is regulated by an opening, which can be more or less completely closed by a sheet-iron door, attached to a regulating bar. There is also an opening in the upper bed, through which the partially roasted charge can be transferred to the lower one. During the process of roasting this opening is closed by an iron plate.

Each bed is provided with a working-door, and a small flue, for the escape of moisture and fumes, connects the upper one with the condensing chambers.

Each charge for this furnace consists of 5 cwts. of finely divided copper matte, 70 lbs. of imperfectly desilverized residues from a preceding operation, and 25 lbs. of lixiviated lumps which have become caked during a previous roasting.

The roasting may be divided into three periods:—

1. The materials are first mixed and then spread on the upper bed, still hot from the previous charge, and are allowed to remain without stirring for about half an hour, in order that they may become perfectly dried. About 5 lbs. of dry and finely powdered brown coal

are now added, and the whole is well worked, with the rake, for about one hour.

As the air, entering by the working-door, passes directly to the flue, the roasting in that portion of the furnace progresses more rapidly than towards the opposite end; consequently at the expiration of a certain time it becomes necessary to change the position of the charge. The material between the working-door and the flue is now turned back toward the further extremity of the hearth, while that which originally occupied the space between the door and the back of the furnace is spread on the hearth between the door and the flue. The mass is then raked for another hour, subsequently again turned, and afterwards raked during two and a quarter hours, by the two workmen in charge of the furnace, alternately. At this period of the operation 25 lbs. of powdered brown coal are added to the charge, with which it is well mixed, and the whole, in a brightly glowing state, is raked through the aperture in the bottom on to the bed beneath. This first stage of the operation of roasting occupies five and a half hours.

2. No fuel is thrown on the grate during the second period; the partially roasted charge is evenly spread over the surface of the red hot lower hearth, where it is continuously raked during about an hour in order to prevent caking. The flue between the furnace and condensing chambers is closed by a damper, in order to prevent a further rise of temperature by the rapid oxidation of sulphur, and the combustion of the brown coal which has been added. In the course of half an hour the brown coal has become entirely consumed, and, after being continuously raked during an hour, the position of the different portions of the charge in the furnace is changed. The damper is now withdrawn and oxidation is accelerated by the admission of air during one hour and a half. From this period the temperature of the mass gradually diminishes, and the charge ultimately assumes a dark colour.

In order to determine the progress of the operation, and to ascertain whether this period should be further prolonged, a sample is taken from different parts of the hearth; this is cooled on a tile, and any lumps it may contain are sifted out. The finely divided powder is spread on an ordinary white plate, and a sufficient amount of water is added to moisten it throughout. If the resulting solution be of a blue colour, and the addition of common salt produces a white precipitate, it indicates that the formation of silver sulphate has commenced, and that the second roasting period may be considered finished. Should the washing of the sample yield a greenish solution, indicating the presence of iron salts, the operation must be further continued.

The second period of roasting usually occupies two and a quarter hours.

3. The fuel employed during the third period should be oak, beech, birch, or some other hard wood, but fir must be avoided, as it produces a smoky flame, exercising a reducing action, resulting in the formation of

cuprous oxide and the separation of metallic silver. The flame rises from the grate to the arch of the furnace, and does not come into direct contact with the charge lying upon the hearth. The draught is regulated by dampers in accordance with the direction of the wind, &c.

The mixture is now thoroughly and continuously worked over by the rake, and at the expiration of an hour it has acquired a dull red-heat, which is afterwards increased to full redness. But few lumps or clots should be formed during the progress of roasting, and these are not broken down but become more compact in proportion as the temperature increases. At the expiration of an hour and a half, that portion of the charge lying nearest the fire-bridge is sufficiently roasted. This is determined by lixiviating a sample, which should afford a solution of a light blue colour and yield a dense precipitate of silver chloride on the addition of solution of common salt. The charge is now turned over, and that portion which was originally furthest from the fire-place is brought to the bridge end of the hearth. The whole is constantly stirred, until a second sample, taken from different parts of the mass, shows that it is ready for drawing. If too much heat is applied during this operation, silver sulphate becomes decomposed, in which case the liquid resulting from washing a sample will be quite free from cupric sulphate, and entirely without colour. The third period of roasting occupies five hours and a half, thus making the total period necessary for the complete working of a charge, on the two beds, thirteen hours and a quarter.

In the newer furnaces the number of roasting beds is increased from two to three and four. The latter, though expensive to build, are considered more advantageous than the smaller sizes. They are heated by gas instead of wood.

The loss of silver experienced during the operation of roasting amounts to about 7 per cent., and above 91·5 per cent. of the silver originally present should be in the form of sulphate in the roasted ore. The fume resulting from the treatment of mattes in the roasting furnace is collected from the flues and condensing chambers, and smelted for copper matte. These mattes are rich in silver.

*Lixiviation and Precipitation.*—On being withdrawn from the furnace the roasted material is cooled to about 87° C., and placed, in charges, each of 5 cwts., in the tubs, A, fig. 219, of which there is a series of ten. These are provided with false bottoms, and with filters constructed in a similar way to those employed for the extraction of silver chloride by hot brine. A leaden tube, *b*, 2 inches in diameter, conveys from 2 to 3 cubic feet of water heated to 87° C. upon the top of the charge, which is covered with oakum or straw for its better distribution over the surface. As soon as the liquors begin to flow from the tap, *c*, inserted beneath the false bottom, that on the pipe, *b*, is closed, and another, communicating with the pipe, *a*, of larger diameter, is opened, by which water, heated to the temperature before indicated, and slightly acidulated with sulphuric acid, is supplied to the lixiviating tub at the rate of about 5½ cubic feet per hour.

The washing is continued until the addition of salt to a sample of the liquors flowing off no longer produces a precipitate of silver chloride. The solution of silver sulphate flows from *c*, into the first compartment, B, of a tank 30 feet long, 2 feet wide, and 1½ feet high. From B the liquors enter the compartment, C, by flowing over a division which does not quite reach the top of the vessel, and are then distributed, by an equal number of taps, into ten precipitating tubs, D, provided with false bottoms. This clarifying box, as well as all the other reservoirs employed, is provided with a float, *d*, indicating the height of the liquid.

Upon the filters of the tubs, D, are placed layers of cement-copper about 3 inches in thickness, above which are laid about twenty copper bars 14 inches long, 5 inches broad, and 1 inch thick. The greater portion of the silver is precipitated in these tubs, and the liquors, on leaving them, are received in the lead-lined trough, E, 15 inches in width, and 6 inches in depth, on the bottom of which is a layer of small pieces of sheet-copper. They then flow into the tubs, F, which have false bottoms, and contain a little granular copper and a few bars of the same metal. The desilverized liquors, which have now a temperature of about 56° C., are conducted by the gutter, *g*, into a leaden reservoir, whence they are pumped into a leaden pan, capable of containing about 70 cubic feet, where they are heated to 87° C., and again employed for lixiviation. Half a pound of sulphuric acid is added to each charge of the leaden pan, and has the effect not only of facilitating the solution of silver sulphate, but also of preventing the separation of basic salts. The precipitated silver is removed from the tubs, D, every twenty-four hours, and the filters are taken out and cleaned once a week.

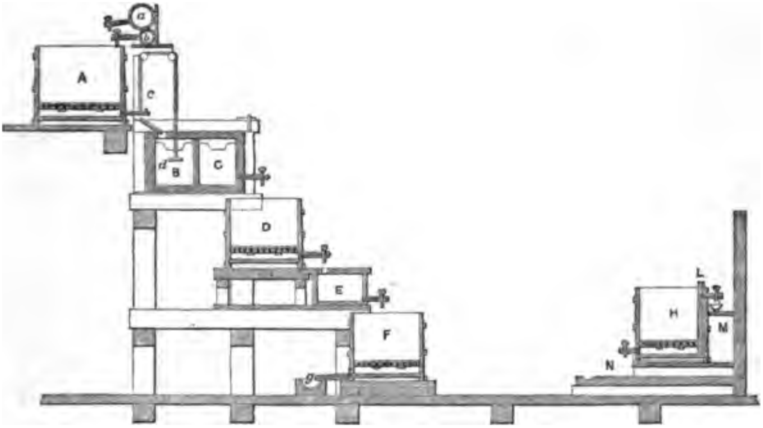

Fig. 219.—Ziervogel's Process; transverse section of plant.

The precipitated silver is chiefly contaminated by the presence of copper and gypsum, and is reduced to powder by being pounded with wooden pestles. It is subsequently lixiviated for several days with sulphuric acid, diluted with eight times its volume of water, in nine tubs, H, in order to remove as much as possible of the remaining copper salts and gypsum, and is finally washed with hot water. The liquors resulting from washing rise through L, and are conducted, by the trough, M, first, over metallic copper, and afterwards into tanks containing scrap-iron; the water from the final washing is run off at N, and conducted, by a gutter, to a lead-lined tank.

The washed silver is moulded into blocks dried in a kiln, and refined in a reverberatory furnace. When the residues are found by assay to contain less than 0·03 per cent. of silver (10 ounces per ton), they are removed, to be treated for copper.

The desilverized liquors are from time to time purified, by throwing down the copper by metallic iron, and the precipitated copper, obtained in the ordinary course of working, is divided into two classes by washing and decantation. The more granular portion is employed for the precipitation of silver, whilst the other, which is contaminated by basic salts, is treated directly for the production of copper.

EXTRACTION IN COLORADO.—At Black Hawk,[1] Colorado, a great variety of pyritic and arsenical ores containing copper, silver, and some gold are treated by a combination of the Ziervogel and Augustin extraction processes. The ores are roasted, partly in heaps and partly in calciners, and smelted, with additions of rich slags, unburnt pyrites, and fluor-spar, to coarse metal containing 25 to 30 per cent. of copper, 600 to 1,000 ounces of silver, and 20 to 30 ounces of gold per ton. This is reduced to a fine powder by grinding and sifting, and roasted for sulphates by two operations. The first, lasting twenty-four hours, is done in a calciner, about 30 feet long, holding 3 tons; and the second in a smaller one taking a charge of 16 cwts. When the copper salts are completely decomposed, the silver sulphate is extracted by boiling water and reduced by cementation with copper in the manner previously described. The residues from the extraction, containing all the gold and about 6 per cent. of the original amount of silver, are mixed with telluride ores rich in gold, and smelted to white metal containing 60 per cent. of copper, 150 ounces of silver, and 50 or 60 ounces of gold per ton. The white metal is subjected to the selecting process of the Welsh smelter (page 427), giving copper bottoms with nearly the whole of the gold, and pimple metal with 75 per cent. copper, 2 ounces of gold, and 150 ounces of silver per ton, which is again selected, giving bottoms that are less auriferous than the first and an argentiferous regulus. The latter is roasted and washed with hot water, but for complete desilverization it is necessary to subject it to a further roasting and extraction with salt.

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[1] This account refers to 1879. The works have since been removed to Denver, where the same process is followed with some modifications.

The final residues are smelted for copper. The treatment of the auriferous bottoms is noticed subsequently under gold.

Von Patera's Process.—Dr. Percy, in 1848, first suggested the extraction of silver from argentiferous ores by its conversion into chloride and subsequent solution in sodium hyposulphite. This paper, which ultimately fell into the hands of Von Patera, resulted, in 1858, in the introduction, at Joachimsthal, of the process now bearing his name. The ores from that district are remarkable for the diversity of their constituents, and in addition to silver contain copper, lead, bismuth, iron, nickel, and cobalt, associated with sulphur, arsenic, and antimony. Mining operations in the vicinity of Joachimsthal, although less productive than formerly, still yield a certain amount of argentiferous ores of extraordinary richness. Those treated by the process under consideration contain, on an average, between 2 and 3 per cent. of silver, and small parcels are sometimes operated on which yield as high as 15 per cent. of this metal. The only fuel to be obtained at a moderate price in the district is lignite, but labour is abundant and moderately cheap.

The extraction of silver from its ores by this process comprehends the following manipulation:—1st. Roasting with common salt, until the silver has been converted into chloride. 2nd. Dissolving out the silver chloride by a cold dilute solution of hyposulphite of sodium. 3rd. Precipitating the silver in the form of sulphide, from its solution in sodium hyposulphite, by the addition of sodium sulphide. 4th. The silver sulphide thus obtained is reduced to the metallic state by exposure in a muffle to a high temperature.

*Roasting.*—The ores, on arriving at the works, if not sufficiently reduced in size, are ground, and then roasted in a furnace, into which superheated steam is introduced.

This apparatus, instead of having the usual long hearth, broad fire-bridge, and wide fire-place, has a hearth 9 feet 6 inches in width, measuring only 6 feet from the bridge end to the take-up of the flue leading to the chimney.

The grate, which is very narrow, is four-fifths of the length of the longer axis of the hearth, from which it is divided by a fire-bridge enclosing an iron tube, protected by clay, and pierced with eight or ten small holes on the side furthest removed from the grate. A charge of 400 lbs. of the ore to be operated on is spread on the hearth of this furnace, and the heat gradually and cautiously raised, in order to avoid agglomeration. No steam is admitted during the first stage of the operation, but as soon as the charge has attained a red heat as much is turned on as can be introduced without materially reducing the temperature. At the expiration of four hours from the time of charging, the ore is withdrawn, and, after being allowed to cool, is ground in a mill to the state of a fine powder, with the addition of from 6 to 12 per cent. of common salt, and from 2 to 3 per cent. of ferrous sulphate.

A charge of this mixture, weighing 3 cwts., is introduced into a furnace similar to that employed for the first roasting, and the second roasting is commenced. The charge is spread evenly over the surface of the hearth, and as soon as a red heat has been attained steam is admitted as before, care being taken, by constant stirring, to prevent agglomeration. The temperature is gradually increased, and at the expiration of from eight to sixteen hours, according to the richness and the composition of the ores, the operation is finished.

The addition of ferrous sulphate to the partially desulphurized ore is for the purpose of effecting the decomposition of common salt in case a sufficient amount of other metallic sulphates should not be produced. The introduction of aqueous vapour is thought to facilitate the chemical decompositions going on in the furnace, and to assist in the condensation of fumes in the flues and chambers prepared for that purpose.

The roasted and finely divided ore, containing silver in the state of chloride, is now taken to the lixiviating shed.

*Lixiviation with Water.*—In addition to chloride of silver, which is insoluble in water, the roasted ores contain variable quantities of copper, zinc, iron, nickel, and cobalt, which, being present chiefly in the form of chlorides and sulphates, are dissolved by washing in a row of tubs, each containing 400 lbs. of roasted ore, hot water being allowed to percolate through the several charges during a period of six hours. By this means the soluble salts are removed, and the liquors passing through the filters are conveyed by a trough, *b* (fig. 220), into a tank, where the metallic oxides are precipitated by lime-water. The precipitate thus obtained is subsequently fused with a mixture of residues and poor ores in a blast-furnace.

The liquors draining into the trough are from time to time tested by sodium sulphide, and as soon as a precipitate is no longer obtained on the addition of a drop of this reagent, the operation is considered finished, and cold water is passed through the tubs for the purpose of lowering the temperature of the residues.

*Lixiviation with Sodium Hyposulphite.*—The residues remaining in the tubs, A, after the removal of the various salts soluble in hot water, are transferred to the tubs, B, which are also provided with filters and false bottoms. At Joachimsthal seven of these are employed, and are placed on a level with the tubs, A (of which there are several), between which and the vessels, B, is a tramway, on which is the bogie, *c*. The tubs, B, stand on trucks, *c*′, which can be run on to the bogie, *c*, and made to traverse, either backwards or forwards, in front of the row of tubs, A. The vessel, B, after receiving a charge of 200 lbs. from the tub, A, is removed to its position on the other side of the tramway, where it is treated with the solution by which the removal of the silver is effected.

This consists of a cold solution of sodium hyposulphite, 1 cubic foot of which is capable of dissolving 0·753 lb. of silver. This is conveyed through the trough, *b′*, and allowed to filter slowly through the mass. In this way the silver chloride is gradually taken up in the form of a double salt, which, passing beneath the false bottom into the trough, *d*, is conveyed to the precipitating tubs.

The duration of this operation is, to a considerable extent, influenced by the richness and composition of the ores, as well as by their state of mechanical division. Parcels containing 15 per cent. of silver are not sufficiently impoverished by lixiviation in less than forty-eight hours, whilst ores containing 1 per cent. can be treated in about twelve hours. Ores which do not contain above 7 per cent. of silver require but one chloridizing and lixiviation, but when richer samples are operated on, two successive roastings and lixiviations become necessary. During the second roasting addition is again made of salt and ferrous sulphate. The lixiviation is considered complete when the liquors dropping from the tubs no longer afford any traces of a precipitate on the addition of a drop of sodium sulphide. The residues are dried and fused with iron pyrites in a blast-furnace.

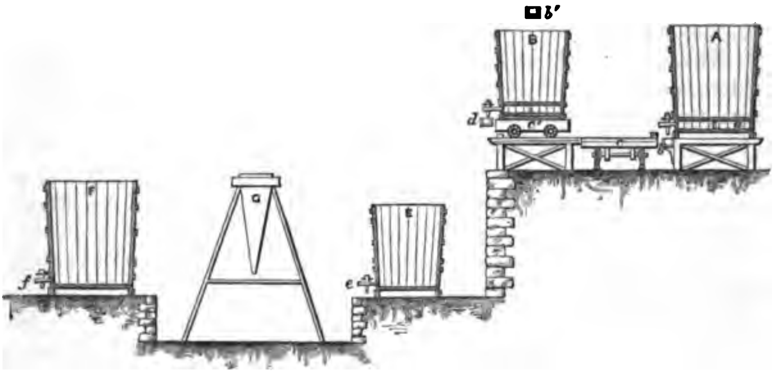

Fig. 220.—Von Patera's Process; transverse section of plant.

*Precipitation of Silver.*—The liquors passing through the filters at the bottom of the tubs, B, are conducted by the trough, *d*, into the vessels, E, F, of which there are ten; six holding 40 gallons each, and four of the capacity of 80 gallons. The precipitant employed is prepared by fusing soda-ash with sulphur, dissolving in water, and boiling the solution with excess of sulphur. The liquor thus obtained, which contains sodium pentasulphide, with a small amount of hyposulphite of sodium, is conveyed to the precipitating vessels in large stoneware jars, and is poured into the argentiferous solutions so long as a precipitate is thrown down on the addition of a further quantity.

The contents of the tubs are first well stirred and then allowed to settle, and a sample of the clear liquid having been taken in a test tube, a few drops of the precipitating solution are added. If a dark-coloured precipitate is the result, it shows that a certain amount of silver still remains in solution, and a further addition of the precipitant is necessary. If, on the contrary, no precipitate takes place, it becomes probable that too large an amount of sodium sulphide may have been added. In order to determine this point, some fresh liquor, holding the double salt of silver in solution, is added to a sample taken from the tub under treatment. Should a precipitate be thus obtained, argentiferous liquor must be cautiously added to the tub until no further reaction takes place. When this point has been attained all doubt as to whether the whole of the silver has been precipitated, on the one hand, and no excess of the precipitant has been employed on the other, is removed by taking two samples of the supernatant liquors, to one of which a few drops of a weak solution of sodium sulphide are added, whilst into the other a small quantity of solution of acetate of lead is introduced. If the addition of sodium sulphide produces no precipitate of silver sulphide, it shows that the whole of that metal has been removed, and should no discolouration take place on the addition of the solution of the lead salt, it indicates that the precipitant has not been added in excess.

The exact neutrality of the residual liquid is necessary in order to obtain satisfactory results, since the liquors from which the silver has been precipitated are employed in the next operation. The presence of an excess of sodium sulphide would evidently convert a portion of the silver into insoluble sulphide, whilst the addition of too small a quantity of this precipitant would leave chloride of silver in solution, and thereby diminish the solvent powers of the liquors for that salt.

Six hours after the addition of the sulphide solution the flocculent precipitate has sufficiently settled to admit of the supernatant liquor being syphoned off into a tank, situated below the level of the floor. It is thence pumped to the level of the trough, *b′*, to be again used in the process of dissolving. The sulphide of silver is drawn off by the taps, *e*, *f*, and placed in a filter-bag of canvas to drain.

Instead of any loss of sodium hyposulphite being experienced during the working of this process, a gradual increase of that salt is the result. This arises from the action of the air on the precipitating liquors, and consequently the solutions employed for dissolving chloride of silver require to be occasionally diluted by the addition of water. The yield by this process amounts to 88 per cent. of the silver present in the ores treated. From 1 to 2 per cent. is finally lost, and the remainder is found in intermediate products. The cost of materials and labour amounted, in 1862, to about 8s. 6d. per cwt. of ore treated.

*Treatment of Silver Sulphide.*—The sulphide of silver, removed from the tubs, E and F, is placed in conical canvas bags, G, supported on wooden frames, and allowed to drain. After standing about half an hour, the bags, with their contents, are placed under a screw-press, and as much as possible of the remaining moisture is expressed. The precipitate is now removed from the bags, dried in a warm room, and afterwards washed in similar filters with warm water. The silver sulphide, thus freed from soluble salts, is again dried and subsequently heated to redness in a muffle to which atmospheric air has free access. In this way the greater portion of the sulphur is burnt off, leaving a residue which contains from 60 to 80 per cent. of silver.

This residue is fused in large graphite crucibles, and any sulphur it may still retain is removed by the addition of metallic iron; the ferruginous sulphide thus produced is skimmed off, and added to the roasting mixture, in place of ferrous sulphate. The surface of the metal is finally cleaned by adding a small quantity of a mixture of bone-ash and wood-ashes, which, on being scraped off, leaves metallic silver of from 940 to 960 fine.

*Residues.*—The extraction residues which contain nickel and cobalt, and are not perfectly desilverized, are, together with the precipitate by lime-water, smelted with additions of low-produce silver ores, 10 per cent. of pyrites, 12 per cent. of slags, and sufficient lime for fluxing. This gives a matte of variable composition, but averaging 17 per cent. of nickel and cobalt, which is re-melted with addition of sodium sulphate, moistened, allowed to fall into powder by exposure to the atmosphere, and subsequently lixiviated to remove soluble salts. The residue, which is rich in silver, is fused with arsenical pyrites for the production of speiss and copper matte. The former is purified by re-melting, and the copper matte is treated for silver and copper as before.

RUSSELL'S PROCESS.—The hyposulphite extraction process, either in the original form prescribed above, or in Kiss's modification, using calcium hyposulphite and sulphide instead of the corresponding sodium compounds, has been largely adopted in the United States and Mexico; but there are certain difficulties in applying it to the treatment of mixed or base ores. When lead is present it becomes partially dissolved, since lead sulphate is soluble in hyposulphite solutions, while if the ore contains copper it will be precipitated with the lead and silver by alkaline sulphides, thus somewhat complicating the subsequent treatment of the resulting product.

According to a recent paper by Mr. C. A. Stetefeldt,[1] a new process, invented by Mr. E. H. Russell, of Park City, Utah, involves important reactions not previously described.

Lead may be completely separated from a sodium hyposulphite solution, in the form of carbonate, by sodium carbonate, without precipitating either silver or copper, which, remaining in solution, are obtained as sulphides, by the addition of sodium sulphide. If calcium sulphide were employed for this purpose, the lime salts contained in the regenerated lixiviation solution would subsequently be precipitated with the lead as carbonates on the addition of sodium carbonate.

The solution of a double hyposulphite of copper and sodium, formed by mixing sodium hyposulphite with sulphate of copper, is stated to exert a powerful decomposing and solvent action upon metallic silver, its sulphides, sulpharsenides, and sulphantimonides, whereas the simple alkaline hyposulphite solution dissolves only chloride of silver. If, therefore, a charge of ore is first washed by a solution of sodium hyposulphite to remove chloride of silver, and afterwards with cuprous hyposulphite, the solution of the latter salt being called by the inventor the "extra solution," an additional amount of silver is obtained, which would have been lost in the tailings if treated by sodium hyposulphite alone.

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[1] 'Proceedings of the American Institute of Mining Engineers,' May 1884.

The process may also be employed to extract silver from unroasted ores, or after they have experienced an oxidizing calcination, without chloridizing.

The mixed sulphides of copper and silver obtained as a precipitate, are dissolved in nitrated sulphuric acid, the escaping nitrous fumes being re-converted into nitric and nitrous acids in the usual way. From the solution silver is precipitated by metallic copper, while copper sulphate is obtained by crystallization. A portion of the latter is used in preparing the "extra solution."

The process has been tested upon a working scale at the Ontario Mill, Utah.

The mixed sulphides, when treated by roasting and fusion, yield silver of from 874 to 876 fine, while a regulus containing from 24 to 27 per cent. of copper, and from 2,700 to 3,000 ounces of silver per ton, is formed to the extent of about 75 per cent. of the weight of the bullion obtained.

By a new method of treatment the mixed sulphides, collected on a filter-press, are dissolved, without previous drying, in sulphuric acid to which sodium nitrate has been added. No heat is required, the reaction being very violent. Gold, if present, is not dissolved, but remains with any silver chloride which may result from the presence of chlorine either in the acid or in the sodium nitrate. Sulphur separates in globules, which are employed in the preparation of sodium sulphide.

The argentiferous solution is drawn off into a tank, where the silver is precipitated by metallic copper. Sulphate of copper is obtained from these liquids by crystallization, and the nitrous gases are recovered by passing them through a coke tower with a spray of water or sulphuric acid. Ores treated by this process are stated to yield from 90 to 94·4 per cent. of the silver they contain.

On comparing it with amalgamation, to which Mr. Stetefeldt regards it as superior both from sanitary and economic points of view, he considers it likely to effect a revolution in the treatment of silver ores, and a diminution of working expenses such as has been rarely witnessed in the history of metallurgy. Statements upon this subject are of a conflicting character, but up to the present time the anticipations expressed above have not been realized.

**Claudet's Process.**—The object of this process is the recovery of the silver which, in the form of chloride, is dissolved in the liquors resulting from the treatment of cuprous pyrites by the wet method of extraction. These contain a large amount of undecomposed common salt, which dissolves the silver chloride, and from which, formerly, it was found impossible to precipitate the silver in a concentrated form.

It has long been known to those engaged in copper-extraction that the copper-precipitate produced from Spanish and Portuguese ores contains not only a notable quantity of silver, but also distinct traces of gold. No successful attempt to separate the precious metals and to turn them to profitable account had, however, been made up to the commencement of the year 1870, when Mr. F. Claudet patented a process for their separation from ordinary copper liquors by the addition of a soluble iodide.

The amount of silver present in burnt ore seldom exceeds 1½ ounce per ton, but as the whole of this is never obtained in solution, it follows that, in order to obtain satisfactory results, in dealing with such minute quantities, the process employed should be both cheap and expeditious.

The vats in which burnt ore, which has been roasted with salt, is lixiviated, generally receive some eight or nine successive washings with water, with weak liquors, and with water acidulated by hydrochloric acid. Of these washings the first three only contain a sufficient amount of silver to be worth working.

For the purpose of removing the soluble salts from the ground and roasted ore, hot water is first employed, and, as a large proportion of the sodium chloride used remains undecomposed, it acts as a solvent for the silver chloride produced during the process of furnacing.

The analysis of a first washing from a copper tank gave Mr. Claudet the following results :—

ANALYSIS OF STRONG LIQUORS.

Sp. Gr. = 1·240.

| | Contents per Gallon.<br>Grains. |
|---|---|
| $Na_2SO_4$ | 10,092 |
| NaCl | 4,474 |
| Cl (combined with other metals) | 4,630 |
| Cu | 3,700[1] |
| Zn | 480 |
| Pb | 40 |
| Fe | 32 |
| Ca | 52 |
| Ag | 3·06 |

As, Sb, Bi, &c., not estimated.

Proportion of Cu to Ag = 10,000 : 8·2.

The respective amounts of copper, chlorine, sulphur, and silver contained per gallon in nine successive washings of one tank of ore are given in the following table :—

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[1] 405 grains of this copper existed in the state of cuprous chloride.

LIQUORS RESULTING FROM NINE WASHINGS OF ONE TANK OF ORE.

| No. of Washing. | Sp. Gr. | No. of Grains per Gallon. | | | |
|---|---|---|---|---|---|
| | | Cu. | Cl. | S. | Ag. |
| 1st . . . . | 1·285 | 5,230 | 10,798 | 1,324 | 4·06 |
| 2nd . . . . | 1·250 | 4,600 | 9,079 | 1,455 | 3·25 |
| 3rd . . . . | 1·175 | 1,935 | 3,215 | 1,881 | 1·05 |
| 4th . . . . | 1·080 | 646 | 717 | 1,255 | 0·19 |
| 5th . . . . | 1·095 | 666 | 643 | 1,436 | 0·12 |
| 6th . . . . | 1·070 | 692 | 544 | 1,588 | 0·06 |
| 7th . . . . | 1·060 | 342 | 217 | 938 | 0·03 |
| 8th . . . . | 1·030 | 200 | ... | 434 | 0·06 |
| 9th . . . . | 1·020 | 117 | ... | 294 | 0·04 |

Washings 1 and 2 contain 82·50 per cent. of total silver.
,, 1, 2 and 3 contain 94·30 ,, ,,

The various operations for the extraction of silver are conducted in the following manner, and as the first three washings contain nearly 95 per cent. of the total amount of that metal dissolved, these alone are treated.

The liquors are first run into wooden cisterns, each having a capacity of about 2,700 gallons, where they are allowed to settle.

*Estimation of Silver in the Liquors.*—The yield of silver per gallon is now ascertained by taking a measured quantity, to which are added hydrochloric acid, potassium iodide, and a solution of acetate of lead. The precipitate thus obtained is thrown upon a filter, and, after being dried, is fused with a flux consisting of a mixture of sodium carbonate, borax, and lamp-black. The resulting argentiferous lead is passed to the cupel, and, from the weight of the button of silver obtained, the amount of that metal in a gallon of liquid is estimated.

*Precipitation of Silver.*—The liquor from the settling vat is allowed to flow into another of slightly larger capacity, whilst at the same time the exact amount of some soluble iodide, necessary to precipitate the silver present, is run into it from a graduated tank, together with a quantity of water equal to about one-tenth of the volume of the copper solution. During the filling of the second tank its contents are constantly stirred, and, when filled, it is allowed to settle during forty-eight hours.

The supernatant liquors are, after being assayed, run off, and the tank is again filled. About once a fortnight the precipitate collected at the bottom is washed into a vessel prepared for its reception.

This precipitate is composed chiefly of a mixture of lead sulphate, lead chloride, silver iodide, and subsalts of copper, from which the latter salts are readily removed by washing with water acidulated by hydrochloric acid. When thus freed from copper salts, the precipitate is decomposed by metallic zinc, which reduces the silver iodide, and the lead chloride. The results of this decomposition are—

First, a precipitate rich in silver, and containing a certain amount of gold.

Second, zinc iodide, which is employed in subsequent operations to precipitate further quantities of silver, when its strength in iodine has been determined.

The more important constituents contained in a sample of the precipitate were estimated with the following results :—

Analysis of Silver Precipitate.

Moisture 25 per cent.

| | | | oz. | dwt. | gr. | |
|---|---|---|---|---|---|---|
| Ag | 12·500 | per cent. = | 4,083 | 0 | 0 | per ton. |
| Au | 0·0965 | = | 31 | 10 | 0 | „ |
| Zn | 1·440 | | | | | |
| Pb | 58·400 | | | | | |
| Cu | 0·600 | | | | | |
| CaO | 1·10 | | | | | |
| Fe | 0·70 | | | | | |
| $SO_3$ | 6·68 | | | | | |
| Insoluble | 7·60 | | | | | |

Mr. Claudet's process for the recovery of silver is now universally adopted by copper-extractors throughout Europe, and annually yields large amounts of silver and gold, which, previous to its discovery, were entirely lost to commerce.

The production of the precious metals by this process in the United Kingdom alone have been as follows :—

| | Silver. | Gold. |
|---|---|---|
| | Ounces. | Ounces. |
| 1870 to 1879 | 492,727 | 1,925·6 |
| 1880 | 206,981 | 1,047·6 |
| 1881 | 258,463 | 1,489·8 |
| 1882 | 377,189 | 1,518·9 |
| 1883 | 356,349 | 1,983·2 |
| To January 31st, 1884 | 1,691,709 | 7,965·1 |
| Estimated 1884 and 1885 | 800,000 | 4,000 |
| | 2,491,709 | 11,965 |

The patent having expired early in 1884 no record has since been kept of the quantities of silver and gold annually extracted, and the amounts since that date are estimations only. The total cost of treating the liquors from 1 ton of cupriferous pyrites by the Claudet process is about 8d., and the value of silver and gold recovered from each ton of Rio Tinto burnt ore is 6s.

## GOLD.

Gold is possessed of a characteristic yellow colour, and is the most malleable of all the metals. One grain of gold may be beaten into a leaf having a superficies of 56 square inches, which, from this measurement, and the known specific gravity of the metal, is calculated to have a thickness of only one 280,000th of an inch.

When in extremely thin leaves, gold is, to a certain extent, translucent, and, on being held between the observer and the light, appears of a greenish colour. When large quantities of gold are fused, and allowed slowly to cool, octahedral crystals, frequently modified on their edges and angles, are obtained. Native gold likewise affords numerous crystals belonging to the cubic system, but in the greater number of these the faces of the octahedron predominate. The specific gravity of gold is 19·50.

Gold fuses at a temperature of about 1,102° C., and when still more strongly heated, affords sensible metallic vapours. If a powerful electric discharge be passed through a fine gold wire it becomes dissipated, and a sheet of white paper held beneath is stained with a purple line caused by a deposit of minutely divided gold. If, instead of a sheet of white paper, a plate of polished silver be employed, it is traversed by a brightly gilded line firmly attached to its surface. A globule of gold, exposed between carbon electrodes to the action of a powerful voltaic current, enters into fusion, and gives off abundant metallic fumes, by which its weight becomes rapidly diminished.

When precipitated from its solutions, gold assumes a dark brown colour, but on being rubbed by a piece of polished steel, or other hard surface, it readily assumes its ordinary colour and metallic aspect. If precipitated gold in this form be strongly heated, and, when in that state, struck with a hammer, its particles become welded and united into a solid mass, without their having undergone actual fusion.

The gold used in the manufacture of jewelry, as well as that employed for being coined into money, is invariably alloyed with some other metal, such as copper, and is therefore never pure. Pure gold may be indefinitely exposed to the action of air and moisture without becoming tarnished, nor is it oxidized by being kept in a state of fusion in open vessels. Neither sulphuric, hydrochloric, nor nitric acids attack gold, even when in a finely divided state; but by aqua regia it is readily attacked, and dissolved in the form of auric chloride. Gold may also be dissolved by hydrochloric acid to which some substance capable of liberating chlorine has been added. Among these may be mentioned chromic acid and peroxide of manganese.

Bromine, even in the cold, attacks this metal, although by iodine it is but sparingly acted on, even by the aid of heat. Gold is not directly attacked by sulphur at any temperature, but when fused with alkaline sulphides, is rapidly acted on with the formation of double sulphides.

## GOLD ORES.

Gold occurs in the metallic state, alloyed with more or less silver, and frequently with minute quantities of copper and iron. It is also occasionally found in combination with the rare metals palladium and rhodium, as well as with mercury, forming a native amalgam. With tellurium it forms several compounds of great mineralogical interest, and of some commercial importance.

NATIVE GOLD.—This generally presents a characteristic yellow colour somewhat paler than that of the refined metal, but its natural surfaces are sometimes dull or tarnished, and require to be rubbed with some hard substance before they assume the ordinary appearance of gold. The hardness of gold is less than that of iron, copper, or silver, but greater than that of either lead or tin. When broken by repeated bendings it presents a matted silky structure, which is more or less fine, in accordance with the purity of the specimen. Native gold occurs crystallized, in branches, filaments, and plates, and in disseminated grains, mixed with, and forming part of, various alluvial deposits. Crystalline specimens are likewise numerous, the octahedron being generally the dominant form. Crystals of gold seldom occur isolated, but are more frequently grouped in irregular branches. Their faces are often dull, and sometimes slightly rounded, even in specimens directly extracted from the vein, which cannot have been exposed to attrition.

The aggregations of gold which sometimes occur in auriferous veins, when closely examined, are often found to consist of a series of minute octahedra, implanted one upon another, so as to form a kind of chain.

The grains and fragments found in alluvial deposits vary greatly in size, but are generally small, constituting "gold dust." When of the size of a nut and upwards they receive the name of "nuggets"; and in some localities such pieces are not of unfrequent occurrence.

A nugget was once discovered in Cabarrus County, North Carolina, weighing 37 lbs. troy. In Paraguay, masses of gold varying from 1 to 50 lbs. in weight, have been obtained. Various lumps varying from 16 to 17 lbs., and one weighing 27 lbs., have been found in the Ural district. In the valley Taschku Targanka a fragment was met with, in 1842, which weighed nearly 100 lbs. This specimen has been deposited in the Museum of Mining Engineers at St. Petersburg. Large specimens of gold have been obtained from California, and very large masses have been found in the Australian diggings. The "Welcome Stranger," found near Dunolly, Victoria, afforded 2,268 ounces of gold.

The composition of native gold, obtained from different localities, is given in the following table, from which it will be seen that it is essentially an alloy of gold and silver. The specific gravity varies from 14·4 to 19·3, being lowest in the most argentiferous kinds. The term *electrum* is sometimes given to native gold in which the proportion of silver is one-third or more. This is of a pale primrose or straw yellow tint.

ANALYSES OF NATIVE GOLD.

| Locality. | Au. | Ag. | Fe. | Cu. | Analyst. |
|---|---|---|---|---|---|
| Transylvania, Vöröspatak | 60·49 | 38·74 | ... | ... | Rose. |
| „ Barbara | 84·80 | 14·68 | 0·13 | 0·04 | „ |
| Beresof | 91·88 | 8·03 | ... | ... | „ |
| Siränovski, Altai | 60·98 | 38·38 | ... | 0·33 | „ |
| Brazil | 94·00 | 5·85 | ... | ... | Darcet. |
| Bolivia, Ancota | 94·73 | 5·23 | 0·04 | ... | Forbes. |
| „ Tipuani | 91·96 | 7·47 | trace | ... | „ |
| N. Grenada, Bogota | 92·00 | 8·00 | ... | ... | Bouss. |
| „ Trinidad | 82·40 | 17·60 | ... | ... | „ |
| Peru, Carabaya | 97·46 | 2·54 | ... | ... | Forbes. |
| „ Yungas | 79·89 | 20·11 | ... | ... | „ |
| Nova Scotia, Tangier | 98·13 | 1·76 | trace | 0·05 | Marsh. |
| California | 90·70 | 8·80 | 0·38 | ... | Rivot. |
| „ | 90·96 | 9·04 | ... | ... | Oswald. |
| Canada, Chaudière | 89·24 | 10·76 | ... | ... | S. Hunt. |
| Australia | 99·28 | 0·44 | 0·20 | 0·07 | Northcote. |
| „ Bathurst | 95·68 | 3·92 | 0·16 | ... | Henry. |
| „ Mitta Mitta | 89·57 | 10·43 | ... | ... | Ward. |
| Tasmania, Giandara | 92·77 | 7·23 | ... | ... | „ |
| „ Black Boy Flat | 94·95 | 4·66 | 0·08 | ... | „ |

According to Dana, the average fineness of the native gold of California is 880 thousandths. Australian gold is purer than Californian, and averages from 900 to 960 thousandths of pure metal. The gold of Canada contains from 10 to 15 per cent. of silver, while that from Nova Scotia is generally very pure.

*Telluride Ores.*—In the few localities producing tellurium it is almost invariably associated with gold and silver, forming several minerals, of which the following are the most important:—

*Petzite* $(AgAu)_2Te$, containing from 18 to 26 per cent. of gold, occurs in Transylvania (Nagyag, Offenbanya, and Rezbanya), California (Calaveras County), and Colorado (Gold Hill district).

*Hessite* is a silver telluride of the same constitution, free from gold.

*Sylvanite or Graphic Tellurium* $(AuAg)Te_2$, is the most abundant tellurium mineral at Nagyag. It occurs in twins and other groups of silvery white crystals, resembling Hebrew characters interspersed in quartz vein stuff. It is also found in the American localities. The proportion of the gold is from 25 to 27 per cent., and of silver 11½ to 13 per cent.

*Calaverite* is a more auriferous variety of a brassy yellow, with gold 39 and silver 33 per cent., found in some quantity at Gold Hill, Colorado.

*Nagyagite, Foliated or Black Tellurium* $(PbAu)(TeS)_2$, is a dark, lead-grey, foliated mineral, somewhat like antimonite, characteristic of the Transylvanian mines. A typical specimen contained—

| Pb. | Au. | Ag. | Cu. | Te. | S. |
|---|---|---|---|---|---|
| 50·95 | 9·10 | 0·53 | 0·99 | 30·09 | 9·70 per cent. |

Sometimes antimony is present up to about 4·5 per cent.

All the telluride ores are fusible at a low heat, and are very easily decomposed by air or alkaline fluxes on charcoal, leaving a residue of gold or auriferous silver.

Bismuth, like tellurium, combines with gold, but the only definite compound described is maldonite ($Au_2Bi$), from Maldon, Victoria. This contains gold 64·5 and bismuth 35·5 per cent. Native bismuth from Australia and South America is often notably auriferous.

Arsenical-, magnetic-, and ordinary iron-pyrites, when present in auriferous veins, almost invariably contain gold, from a few grains to a few ounces per ton, and often constitute the most valuable portions of such veins. A trace of gold may be found in galena by operating on a large quantity, but the lead smelted from it is rarely auriferous unless pyritic and antimonial ores accompany it, and in such cases the gold goes rather with the copper than with the lead.

## DISTRIBUTION OF GOLD.

Gold appears to be as widely distributed as the other metals, but it usually occurs in such minute quantities that it may escape observation even when sufficient is present to pay the cost of extraction.

Native gold, *in situ*, is most frequently met with in quartz veins intersecting metamorphic rocks, and is, almost invariably, associated with iron pyrites and other metallic sulphides, such as galena, blende, &c. The rocks enclosing gold veins are mostly chloritic, talcose, and argillaceous slates, and, less frequently, they are met with in mica schist, hornblendic slates, gneiss, diorite, or porphyry. Auriferous veins also occur in granite. Itacolumite, a laminated talcose quartzite, is common in some gold regions, in Brazil and North Carolina, and schists containing specular iron or granular magnetite sometimes contain gold.

The gold of quartz veins occurs in the form of plates, strings, and thin scales, as well as in crystalline grains; these are frequently apparent to the eye, but rock showing no visible traces of gold is often sufficiently rich to admit of being treated with profitable results.

It is not, however, from the treatment of auriferous quartz that the gold of commerce has been mainly derived; a very large proportion of it having been obtained from alluvial diggings, in which gold is separated from its associated sands and gravels by washing.

In such deposits nature has, on a vast scale, performed the operations of crushing and concentration, and has finally deposited the gold in positions from which it can be cheaply extracted. To this circumstance are mainly attributable the sudden fluctuations which have, from time to time, taken place in the gold-production of the world. On the discovery of a new gold-region a large amount of unskilled labour is at once applied to the extraction of gold from alluvial diggings, while to obtain the same amount of gold from quartz veins would necessitate skilled labour and the expenditure of a large amount of time and money. Indeed, had not this natural disintegration and concentration taken place, a large portion of the gold annually collected could not be advantageously brought into the market. In California the auriferous gravel-beds are of vast extent, and have sometimes a thickness of 250 feet.

Alluvial gold occurs in the form of flattened grains or scales of different degrees of fineness, the size depending partly on its original form of occurrence, and partly on the distance to which it has been transported by water. Gold is widely distributed over the surface of the globe, occurring in rocks of various ages, from the oldest almost to the most recent. But although this metal occurs in many regions of crystalline schists, it exists in comparatively few localities in sufficient quantities to render its extraction remunerative.

In Cornwall and Devon tin-streams afford occasional specimens of gold, but not in sufficient quantities to make its collection of commercial importance. The older schistose rocks of North Wales, and particularly those of Merionethshire, have long been known to enclose auriferous veins. This gold-bearing district appears, however, to be confined to an area of about twenty-five square miles, chiefly lying north of the road leading from Dolgelly to Barmouth.

It has been ascertained that many of the quartz veins occurring in this neighbourhood contain gold, but the amount found has always proved insufficient to pay working expenses. In 1861, nearly 3,000 ounces were obtained from pockets at the Vigra and Clogau mines; and this result, having become known, led to the exploration of nearly all the quartz veins of the district. These operations were carried on, with declining activity, during some four or five years, by the end of which period those embarked in the enterprise had generally arrived at the conclusion that gold-mining in Wales had not been remunerative. The total quantity of gold produced from the commencement of operations in North Wales to the 1st of April 1866, amounted to 12,800 ounces, of which Vigra and Clogau produced 11,778 ounces. The chief portion of the gold in these mines was met with in the form of a short deposit, which soon became exhausted, but the Clogau Mine, although long since unproductive, is still in operation.

In Scotland, gold occurs at Leadhills in Lanarkshire, and at Glen Coich in Perthshire, but in small quantities only, although in the reign of Elizabeth extensive washings for gold were carried on in the alluviums of Leadhills. More recently gold has been discovered in Sutherlandshire, but the district, after numerous trials by returned Australian and Californian miners, has not been proved to be productive on a large scale.

Towards the close of the last century gold was discovered in the county of Wicklow, disseminated in quartzose and ferruginous sands. The gold was found chiefly in the form of nuggets of considerable size, and one was obtained weighing 22 ounces. For a short time this gold was collected on a comparatively large scale by the neighbouring peasantry, who, in the course of two months, gathered an amount for which £10,000 sterling is said to have been paid. The working of the deposit was subsequently undertaken by Government, but the supply soon became exhausted, and, after having been about two years in operation, with unprofitable results, the works were abandoned.

France possesses no gold mines, but the sands of some of her rivers are, to a small extent, auriferous. The only quartz veins which have been known to contain gold are that of La Gardette, in the Department of Isère, some veins near Massiac, Cantal, containing a little gold in arsenical pyrites, and a similar vein near Pontgibaud, Puy-de-Dôme. The Rhone and several other French rivers have produced small quantities of gold, and the Ariège is said to have derived its name from the amount of auriferous sands it deposited.

The Rhine has, for centuries, produced small quantities of gold, its sands having been extensively worked in the neighbourhood of Strasburg, &c. In 1846, M. Daubrée made a report to the Academy of Sciences, in which he states that the most productive gravels were those deposited below sandbanks or gravel islands which had become eroded by the action of the river, and that gold was found in a somewhat concentrated state only in the coarser gravels, from which the finer sands had been removed by running water. The yield of the year 1846 was estimated by M. Daubrée at £1,800, the washers earning, on an average, from one and a half to two francs per diem.

Gold mines were successively worked in Spain by the Phœnicians, Romans, and Moors, and, although the amount now obtained is insignificant, it is stated to have, at one period, produced large quantities of the precious metal. The present small yield is derived from washing the sands of streams and rivers, and the total annual value obtained may be estimated at about £1,500; but some new alluvial gravel workings have lately been undertaken in Leon and elsewhere.

A number of localities in Italy were known to the ancients as producing gold, and at one period this metal was worked so extensively that the quantity produced is said to have caused a reduction of one-third in its price. At present, the only gold mines of any importance are in the north. The amalgamation works are situated on small streams near the foot of Monte Rosa, where gold is found in the valleys of Anzasca, Toppa, and Antrona. The principal mines are those of Val Toppa and Pestarena, which are being worked by English capitalists, and where the ores consist of a compact auriferous pyrites. The aggregate produce of these mines during the year 1881 amounted to 7,249 ounces, of the total value of £25,562, 6s. 3d.

The amount of gold produced in Germany is small, although in some localities, chiefly on the Rhine and its tributaries, washing and mining operations in pursuit of this metal have been carried on from remote antiquity.

In Tyrol and Salzburg a little gold has long been obtained by the treatment of exceedingly poor ores. At Zell, in 1847, the average yield of the vein-stuff treated was 2½ dwts. per ton. The annual production of the mines of Tyrol and Salzburg may be estimated at 100 lbs. troy.

The most important gold mines of Europe are those of Hungary and Transylvania, where gold is found in veins of auriferous pyrites associated with galena and sulphides of silver. The mines of Hungary have been worked since the eighth century, the operations being conducted with much skill and economy. At Schemnitz and Kremnitz the ores afford both silver and gold, together with a sufficient amount of galena to materially assist in their metallurgical treatment. In Transylvania some of the mines afford the rare combinations of gold and tellurium. The production of gold from the mines of the Austro-Hungarian Empire amounted in 1876 to 5,100 lbs. troy.

In Sweden there is a mine at Edelfors, in Småland, where gold occurs in auriferous pyrites, but the produce of the country is insignificant.

The gold mines of the Russian Empire are situated partly on the eastern flank of the Urals and partly in the interior of Siberia.

The produce of the Russian washings in 1877 amounted to 106,600 lbs. troy.

Gold is found in the rivers of Syria and in parts of Asia Minor, and the Pactolus, a river of Lydia, was anciently celebrated for its golden sands. Gold mines are also known to be worked in Tibet, where this metal is found in quartz veins traversing a crumbling granite. It is also met with in the southern part of Hindostan, where numerous mines have been started of late years, as well as in the Eastern Archipelago. Gold-washing is likewise carried on in China and Japan, both of which countries are believed to afford a somewhat considerable annual yield.

Africa was probably the source of a large proportion of the gold possessed by the ancients, and nearly all modern travellers who have penetrated into the interior of that continent agree in their accounts of its wealth. The whole of the gold which Africa now supplies is in the form of dust and water-worn grains, obtained from alluvial washings.

The gold of Southern Abyssinia occurs in quartz enclosed in granite, and is associated with hæmatite and iron pyrites; gold is also found in alluvial deposits of an ochreous character.

Gold-fields have been discovered in Cape Colony, near the frontier of the Transvaal Republic, which also contains auriferous deposits of importance; but no practical results, on any extensive scale, appear to have been obtained.

The annual amount of gold annually furnished by Africa is estimated at about 3,800 lbs. troy.

The gold-fields of the United States of America may be divided into three great geographical divisions, viz., those of the Atlantic slope, or the Appalachian gold-region, worked to some extent for the last sixty years; those of the Pacific States, of which those of California may be considered among the most important; and those of the Rocky Mountains.

The Appalachian gold-fields are included within the States of Virginia, North Carolina, South Carolina, Georgia, Tennessee, and Alabama, although some others have occasionally afforded specimens of the precious metal. The first notice of the discoveryof gold occurs in Jefferson's 'Notes on Virginia;' Drayton, in his 'View of South Carolina' (1802), also mentions the finding of a small piece of this metal on Paris Mountain.

The first United States gold was coined in 1825, and from that time up to 1830 four-fiths of the gold coinage of the country was of American gold. From 1804 to 1827 North Carolina furnished the whole of the gold produced in the United States, amounting to $110,000; but in 1829 Virginia contributed $2,500, and in the same year South Carolina yielded $3,500. In 1830 Georgia made its first deposit at the Mint, amounting to $212,000. Previous to 1825, all the gold of North Carolina had been procured from washings, but in that year auriferous veins were discovered. This turned attention from the "deposit mines" to "vein mines," and led to the discovery of gold veins in various localities in that and the adjoining States.

In 1852–53, the discoveries which had then been recently made in California, produced great excitement with regard to gold-mining generally, and attention was, as a natural consequence, directed to the auriferous districts of the southern States. Many English and American companies were formed for the purpose of working the mines of the Atlantic coast, and for a time mining operations were actively prosecuted, but without permanently satisfactory results, and the southern gold-region, after a short period of spasmodic activity, again subsided into comparative obscurity.

The Pacific divisions comprehend California, Nevada, Utah, Arizona, Idaho, Oregon, Washington, and Alaska.

Although it had long been known that gold had occasionally been found in California, yet until the acquisition of that region by the United States of America but little was known either of the country or of its productions. The first practical discovery of this metal was made either late in February, or early in March, 1848, by Colonel Sutter, on the south branch of the American River, at a place now called Coloma, and in the following July 4,000 people were employed in washing on the American River and its tributaries, and were extracting gold of the value of from $30,000 to $40,000 daily.

The fame of these discoveries became widely spread during the latter months of 1848 and the spring of 1849, and a rapid influx of immigration commenced, which has, with less activity, continued to the present time, and had in 1874 already resulted in an addition of at least 3,500,000 lbs. troy of gold, value £175,000,000, to the available bullion of the world.

The great auriferous belt of California lies on the western slope of the Sierra Nevada, beginning in the vicinity of the Téjon Pass, and extending through the State to its northern limit. The principal gold region may, however, be said to occupy the western portions of the counties of Mariposa, Tuolumne, Calaveras, Amador, El Dorado, Placer, Nevada, Sierra, and Plumas, with portions of the eastern sides of Yuba and Butte counties.

The rocks of the auriferous belt are principally Jurassic and Triassic, and nothing older than Carboniferous has been found in the gold region.

The gold of California is derived from three sources:—

1st. From auriferous veins, most frequently inclosed in metamorphic slates.

2nd. From deposits of alluvial gold occupying ancient river-courses.

3rd. From deposits in which the gold of ancient river-systems has been redistributed by more modern streams.

Auriferous veins, like all others, are exceedingly variable, not only in dimensions, but also in productiveness. It is, however, generally observed that the widest veins are not usually the richest, and that some of the bands running parallel with the inclosing walls are uniformly more productive than others. As a general rule those veins are most auriferous which contain a considerable amount of metallic sulphides, more particularly iron pyrites, and but few which do not afford a notable percentage of this mineral are found permanently remunerative.

Contrary to an opinion generally entertained a few years since, gold veins are not found to be more liable to impoverishment in depth than are other lodes, some of those on the Pacific coast having been worked, on their inclination, to a depth of more than 1,450 feet without experiencing any diminution in yield.

In many localities, and particularly between the south and middle forks of the Yuba River, the auriferous gravels belonging to the second class have, under ordinary circumstances, a thickness of 120 feet; and when, as is sometimes the case, these ancient river-beds have been protected by a capping of basalt, their thickness sometimes exceeds 250 feet. These deposits are worked by a process known as hydraulic mining, and at one period yielded one-half of the gold annually produced in the country.

These ancient river-beds, or *deep placers*, are of Pliocene age, and frequently enclose trunks of large trees which have become completely silicified.

The attention of the first miners was exclusively directed to deposits of the third class, or *shallow placers*, in which gold lay near the surface and within the reach of those who, without capital, were in possession of ordinary mining tools. These shallow diggings, however, became gradually exhausted, and at the expiration of some five or six years attention was generally directed to deeper deposits.

The greatest exportation of gold from California took place in the year 1852–53, when the amount was estimated at about 240,000 lbs. troy; from that period the export of gold bullion has been gradually diminishing, and the present annual production of the State is not above 70,000 lbs.

The division of the Rocky Mountains, which yields about 23 per cent. of the gold annually produced in the United States, comprehends Colorado, Dakota, Montana, New Mexico, and Wyoming. The alluvial deposits of these states are of less importance than their vein mines, which are often pyritic, yielding smelting ores in large quantities.

The total production of gold in the United States of America during the year 1884 was 124,097 lbs. troy.

Although the most productive gold-fields of North America are comprehended within the limits of the United States, several provinces of the Dominion of Canada annually afford a certain amount of gold.

The existence of this metal in Quebec first attracted attention in 1847, although it is stated that a French Canadian had found specimens of some value long before that date. In 1850 gold was discovered in the alluvium of the Chaudière and various neighbouring streams, which were subsequently worked by the Chaudière Gold-Mining Company; but the returns obtained, both from the alluvial washings and from quartz-mining operations, still remain exceedingly small.

The working of auriferous arsenical pyrites has latterly been commenced at Marmora in Ontario.

The late Sir James Douglas, when governor of Vancouver's Island in 1856, reported the discovery of gold in British territory north of latitude 49°, but stated that in consequence of the hostile attitude assumed by Indians the number of diggers was small. In 1858, however, the stream of immigration set in with sufficient force to overcome the opposition of the natives, and from that period British Columbia has permanently become a gold-producing country.

It has, however, been found, that although the country is rich in gold, the winters are so severe as to preclude the possibility of continuous mining during the colder months, and this, together with the difficulties of access and transport, has caused British Columbia to be somewhat unpopular among gold miners. Latterly the prospect has been improved by the completion of the Canadian Pacific Railway. Nearly the whole of the gold produced finds its way to San Francisco, and as a large proportion reaches that city through private hands it is impossible to obtain exact statistics. The total annual yield of British Columbia is, however, probably between 10,000 and 12,000 lbs. troy.

A few years since attention was called to the Province of Nova Scotia by an article published in 'Blackwood's Magazine,' in which it was stated that gold would be found in the hills south of Annapolis; and a comparison was instituted between that locality and the valley of the Sacramento. So long ago also as 1855 Dr. Dawson, in his 'Acadian Geology,' when describing some of the metamorphic rocks of the country, observes:—"Quartz veins, however, occur abundantly in some parts of this district, and it would not be wonderful if some of them should be found to be auriferous."

In the month of March, 1861, a man who was stooping to drink at a rivulet observed a piece of gold among the pebbles at the bottom. During the course of the following June gold was discovered in quartz veins in the cliff near Lunenburg, and subsequently in the sands on the beach beneath the headland, in which auriferous veinstone had been previously met with. Gold discoveries now followed each other in rapid succession at Lawrencetown, Dartmouth, Sheet Harbour, Isaacs Harbour, Sherbrooke, Waverley, Oldham, and elsewhere.

Auriferous drift occurs in Nova Scotia only in patches of very limited extent, and almost the whole of the gold hitherto obtained has been the produce of the treatment of gold quartz. The veins are generally small, but tolerably rich in gold. The production of gold in this Province in 1883 was 1,175 lbs. troy.

Mexico, although rich in silver, yields comparatively little gold, almost the whole of the latter metal being separated from argentiferous ores.

Little is known respecting the auriferous districts of Central America, except that Costa Rica and some other States annually produce a certain amount of this metal.

The principal mines of Brazil are situated in the province of Minas Geraes, and are wrought chiefly on extensive deposits of auriferous pyrites. The Morro Velho mine, belonging to the St. John d'El Rey Company, was, up to within the last fifteen years, the largest and most prosperous mine in Brazil, having, since 1839, produced a net profit to the proprietors of over a million sterling. This undertaking, after suffering from the effects of a disastrous fire, has lately been stopped by extensive falls of rock underground. Brazil afforded its largest yield of gold about the middle of the eighteenth century, before the comparative exhaustion of its rich alluvial deposits, when the amount on which the royal fifth was annually paid varied from 17,000 to 21,500 lbs. The production of Minas Geraes, by far the most productive mining district of the country, was estimated in 1879 at 4,900 lbs. troy per annum.

The present annual production of the great South American gold-fields north of the river Amazon, comprehending the United States of Columbia, Venezuela, Dutch and French Guiana, is about 27,000 lbs. troy, the Venezuelan mines being the most important.

The first authenticated discovery of gold in Australia was made in 1839 by Count Strzelecki, and the presence of the precious metal was again noticed in 1841 by the Rev. W. B. Clarke, of Sydney; but the existence of extensive gold-fields, almost throughout Australia, was first announced in 1851 by Mr. E. H. Hargreaves, a returned Californian miner.

It would appear from the investigations of Mr. A. R. C. Selwyn that the gold-bearing strata of Victoria are of greater age than the auriferous rocks of California, and that they most frequently belong to the Silurian epoch.

With regard to the occurrence of the precious metal in Victoria, Mr.

R. B. Smyth, then Secretary of Mines for the Colony, remarks ('Intercolonial Exhibition, 1861'): "Gold is now found to occur not only in quartz veins and the alluvial deposits derived from these and the surrounding rocks, but also in the clay stone itself; and, contrary to expectation, flat bands of auriferous quartz have been discovered in dykes of diorite, which intersects the Upper Silurian or Lower Devonian rocks. Quartz of extraordinary richness has been obtained from these bands, and the new experience of the miner is leading him to look for gold in places hitherto entirely neglected."

In Australia, as in California, the gold first obtained was entirely derived from the more recent gravels, but ancient river-beds, often covered by a thick capping of basalt, have since proved highly productive. As in California, quartz veins are now advantageously worked to very considerable depths below the surface.

The largest gold-yield of Victoria was in the year 1856, when the total produce was 249,000 lbs. troy; its annual production is, at the present time, about 72,100 lbs. troy, worth £3,458,440. The total amount of gold produced in the colony, from 1851 to the present time, is estimated at about 4,200,000 lbs. troy; worth approximately, £201,300,000.

The produce of the gold-fields of New South Wales, from their discovery in 1851 to 1882, though less than that of Victoria, was 780,470 lbs., representing a value of £34,870,360.

South Australia has produced small quantities of gold, and Tasmania annually yields about 4,800 lbs. troy, worth £217,000.

In 1882 Queensland produced 19,175 lbs. of gold, of the value of £829,655. In this colony the veins produce both free milling quartz and very refractory pyritic ores, the latter being often very rich in gold, and many new mines have lately been commenced.

Gold was first discovered in New Zealand in 1842, and the principal portion of that which has been exported from the colony has been the produce of the South Island. The yield of New Zealand in 1882 was 24,436 lbs. troy of gold, value £1,170,520.

Various other localities annually afford small amounts of this metal, either from alluvial washings or from regular mining, but the total aggregate yield of such operations is comparatively small. The total value of the gold now annually produced in the world is about £21,000,000.

## ASSAY OF AURIFEROUS MINERALS.

Minerals containing gold are assayed in the same way as ores of silver, but as the former usually contain only a very small proportion of the precious metal, it becomes necessary to operate on larger samples than are used in silver assaying. When these minerals contain lead they may be conveniently fused with a proper quantity of black flux; but ores free from lead, and containing metallic oxides, may be advantageously assayed by the addition of a mixture of litharge and powdered charcoal or black flux. When chiefly composed of siliceous and earthy matters mixed with metallic sulphides, such as mispickel, or iron or copper pyrites, their fusion may be effected by the use of litharge only. Lastly, when these sulphides so preponderate as to yield too large a button of lead for convenient cupellation, a mixture of litharge and nitre may be used with advantage.

ASSAY OF GOLD QUARTZ, ETC.—To make an assay of auriferous quartz, the sample must be finely pulverized, and may be mixed with red lead or litharge, together with a little carbonate of sodium, borax, and an amount of pounded charcoal, sufficient for the production of a button of lead of a convenient size for cupellation.

In the case of very poor ores, the silver derived from the oxide of lead will frequently be sufficient for the purpose of inquartation, while for richer ores, the addition of a little pure silver, at the time of placing the button on the cupel, becomes necessary.

If, besides gold, the ore contains pyrites, or any other sulphuretted mineral, the addition of a reducing agent, such as charcoal, may sometimes be dispensed with, and the fusion be made either with oxide of lead alone or with oxide of lead and a little borax.

When pyrites, or any other metallic sulphide, is present in large quantities, the sample should be first roasted until all traces of sulphur have ceased to be evolved, and then treated as in the case of substances not containing that body, but with the addition of a larger proportion of charcoal and borax. Unless the sulphur is expelled either by a preliminary roasting, or oxidized during the operation, it may give rise to the formation of alkaline and other sulphides, which are liable to cause a portion of the gold to combine with the slag.

*Sampling.*—Although it is easy to estimate with considerable accuracy the amount of gold contained in a sample of ore, it is much more difficult to obtain a fair average sample of the total produce of a vein. When the metal is in a state of fine division, and uniformly disseminated throughout the matrix, this presents little difficulty; but when, on the contrary, it is granular, and occurs in irregular grains, much care is necessary to ensure trustworthy assays.

It is of importance, therefore, that whenever ores are to be assayed for gold, great care should be taken in procuring the samples on which the operation is conducted. With this view the heaps should be well cut through, and about 2 tons taken out of each. The ore thus obtained must be reduced to fragments of the size of beans, which, when a stonebreaker is not at hand, may be accomplished with hammers on an iron plate.

The ore thus prepared must now be thoroughly mixed, made into a heap, and again cut through, taking out of it this time 3 or 4 cwts., which are reduced to the state of coarse powder, in a crushing mill, in a large mortar, or on an iron plate. After being again mixed, the powdered ore is cut through, and about 20 lbs. taken for the purpose of being still further reduced in size; this must be passed through a sieve of fine wire gauze. On the sample thus prepared, five or six different assays are to be made, and the mean of the results is taken as the produce of the ore examined. Any flattened grains of gold which may remain upon the sieve must be carefully collected, cupelled with lead foil, parted, and weighed; the proper proportion of its weight being added by calculation, to the result of the assays. By operating as above described, great accuracy is insured; but when a less degree of exactitude is sufficient, the quantities of ore crushed may be reduced, and the number of assays fewer.

*Fusion with Litharge, Sodium Carbonate, &c.*—When the quartz contains only traces of pyrites, or of any other sulphide, 1,000 grains of finely-divided ore may be mixed with three times its weight of litharge or red lead, 200 grains of sodium carbonate, 200 grains of borax, and from 15 to 20 grains of pulverized charcoal. Auriferous pyrites should be first pulverized and then roasted, until all odour of sulphur has ceased to be evolved. The roasted ore is mixed with half its weight of dry sodium carbonate, twice its weight of red lead or litharge, a proper amount of charcoal, and some fused borax. The mixture must be introduced into an earthen crucible, of which it should not occupy more than one-half the capacity, and after being thoroughly fused in an assay furnace the pot and its contents are removed by proper tongs and allowed to cool. When sufficiently cold, the crucible is broken, and the button of lead removed for the purpose of being cupelled. If, in the first experiment, the button obtained weighs less than 100 grains, a little more charcoal ought to be added. Instead of breaking the pot, the crucible, on being removed from the fire, may be held between the bent jaws of a pair of tongs and its contents poured into a conical cast-iron mould.

*Fusion with Red Lead or Litharge.*—In cases where the sample of quartz contains a sufficient amount of pyrites to reduce a quantity of lead convenient for cupellation, the assay may be effected by fusion with litharge, or red lead alone. When this method is employed, oxide of lead must be used in large excess, and 1,000 grains of the ore may be fused with from four to five times its weight of red lead or litharge. If the button obtained be not sufficiently large, its size may be increased, to any desired extent, by the judicious addition of lamp-black or powdered charcoal.

*Cupellation.*—The buttons of alloy obtained are cupelled, with precautions enumerated when treating of the assay of alloys of silver, although when gold is the metal sought, the process is in a slight degree varied.

When the resulting button consists of an alloy of lead, silver, and gold, with a small admixture of more oxidizable metals, its cupellation presents even less difficulty than in the case of alloys of lead and silver, because in the first place gold is less volatile than silver, and consequently may be exposed to a greater heat, and in the second less loss is experienced at a high temperature by absorption into the cupel.

When in addition to gold, silver, and lead, the assay button contains copper, it must be cupelled, like similar alloys of silver; but as copper possesses a greater affinity for gold than it has for silver, a large addition of lead must be made to insure the production of a pure button.

*Parting.*—When, as is almost universally the case, the button obtained by the fusion of the ore contains, in addition to lead and gold a notable proportion of silver, it must be cupelled at a moderate temperature, and, if necessary, an additional quantity of silver added. By operating in this way, the button obtained on the cupel consists of an alloy of silver and gold, which is afterwards treated with an excess of nitric acid. This effects the solution of the silver, and leaves the gold in the form of a brown powder, in the bottom of the flask in which the operation has been conducted. In order to obtain exact results, it is, however, necessary that a certain relation should exist between the amount of the two metals of which the alloy is composed, since if the silver be not present in sufficient quantity the mixture is not completely attacked by nitric acid. On the other hand, when too large a proportion of this metal is added, the gold remains in a pulverulent form, which renders its collection for weighing somewhat difficult.

The above-described operation, which has received the name of "parting," succeeds best when the alloy contains about three parts of silver to one of gold, and therefore, in all cases, the addition of silver must be so managed as to agree as closely as possible with this proportion. If the alloy contains less than two and a half parts of silver to one of gold, the solution of the silver cannot be completely effected.

The operation of adding the proper amount of silver to an alloy to reduce it to the right standard for the process of parting is called "inquartation." The inquartated button should be carefully flattened with a polished hammer on a steel anvil, and afterwards attacked in a small flask or large test tube by nitric acid of specific gravity 1·18. After having been boiled for ten minutes with acid of this strength, the liquid is carefully poured off, and the residue heated to ebullition, during a few minutes, in acid of specific gravity 1·28. The acid is now carefully decanted, and the residual gold, after being washed with distilled water, is transferred to a thin porcelain capsule, from which the water is, as far as possible, removed, and the remainder evaporated in a water-bath. When freed from moisture, the pulverulent gold, after heating to redness, may either be weighed directly in the capsule in which it has been dried, or be folded in a little lead-foil and again passed to the cupel, when it is obtained in the form of a metallic globule.

If, in addition to gold, the mineral contains silver, and it be desirable to ascertain its amount, it is necessary to first cupel the button of lead without addition of silver. The alloy thus obtained is weighed, and its weight noted, deduction being made for the amount of silver derived from the reduced oxide of lead, which must be ascertained by experiment. It is also necessary to examine the red lead, or litharge, in order to ascertain if it contains traces of gold, and in case of that metal being likewise present, due allowance for the amount found must be made on the produce obtained. If the silver be not sufficient for the purposes of parting, more is added, by folding the bead, with a bit of pure silver, in a piece of lead free from the precious metals, and again cupelling. Lastly, the alloy obtained is treated with nitric acid, and the amount of gold present determined by weighing.

The weight of silver contained in the ore will evidently be represented by that of the button of alloy from the first cupellation, less the united weights of the gold in the ore, and of the silver and gold (if any) afforded by the reduced oxide of lead.

It may be observed, that when proper precautions are taken to obtain a fair average sample, and the mean of a sufficient number of assays is taken, there is no difficulty in ascertaining, with a considerable degree of accuracy, the yield of auriferous quartz.

## Assay of Gold Bullion.

The composition of commercial gold bullion varies within wide limits, some being nearly pure gold, while other kinds may contain half their weight or more of other metals, especially of silver and copper. Silver bullion also often contains gold, from a few grains to the troy pound upwards, constituting the so-called *doré* or parting silver, and even such very small quantities of the more precious metal may be of commercial value, if the bullion can be employed for refining purposes instead of chemically pure silver, when the gold is recovered without extra cost. In all cases, therefore, it is essential that bullion containing gold should be very carefully assayed. This is done by cupellation, inquartation, and parting in the manner already described for the assay of ores, the processes being sometimes modified in minor details according to the composition of the material under examination.

Generally half a gramme of the alloy is carefully weighed and subjected to cupellation with a proper quantity of lead, together with an amount of pure silver equal to about three times the weight of the gold supposed to be present. Doré silver, when it contains copper, is cupelled with lead in the same way as silver bullion, before parting; but plate or cake silver that has been refined from lead, and is practically pure, is dissolved in nitric acid without cupellation. As the quantity of gold contained is often very small, larger samples are required than for the assay of gold bars.

The amount of lead necessary for passing an alloy of gold on the cupel depends on the quantity of copper which may be present. The following proportions have, after careful experiment, been generally adopted :—

| Amount of Gold in Alloy. | Amount of Lead necessary for Cupellation. |
|---|---|
| 1,000 | 1 part |
| 900 | 10 parts |
| 800 | 16 „ |
| 700 | 22 „ |
| 600 | 24 „ |
| 500 | 26 „ |
| 400<br>300<br>200<br>100 | 34 „ |

In the case of ordinary gold bars from alluvial and quartz mines, in which the proportion of copper is invariably small, the alloy being chiefly silver, the half gramme employed for assay is passed to the cupel with 2 grammes only of lead. A piece of copper wire weighing 15 milligrammes is also often added, which, by preventing the silver button from vegetating on the cupel, ensures its malleability.

The button removed from the cupel is squeezed, laterally, between the jaws of a pair of strong pliers to loosen any adhering litharge, which is removed by the aid of a stiff brush, and is afterwards flattened, by a smooth-faced hammer on a polished anvil, into an elongated disc about ¾ of an inch in length. This, after being annealed, is passed repeatedly through a flatting-mill until it has assumed the form of a thin strip from 2½ to 3 inches in length, which is again annealed, and coiled by rolling between the finger and thumb. The coil or *cornet* is now introduced into a long-necked flask containing about an ounce of pure nitric acid of 22° Baumé = 1·18 sp. gr., and boiled until red fumes have ceased to be given off. This acid is poured off, and the cornet again twice boiled, each time for about ten minutes, in acid of 32° Baumé = 1·28 sp. gr. In the last two boilings a piece of charcoal, consisting of half of a charred pea, is introduced for the purpose of preventing ebullition from taking place irregularly, and with explosions, by which the cornet might be broken, or a portion of the liquid projected out of the flask. The acid is poured off, and, after the third attack, the cornet is twice washed with distilled water. The flask is now filled with water and reversed into a small crucible of fine clay covering its neck; by this means the cornet is deposited gently, and without breaking, in the bottom of the crucible, and the water which covers it is poured off. The crucible and its contents are heated to redness in the muffle, care being taken to avoid the fusion of the gold. From the weight of the cornet obtained, the fineness of the alloy is calculated, but in all cases where great accuracy is required at least two assays are made of each bar.

After boiling in nitric acid, the cornet is of a brownish-yellow colour, of a spongy texture, and exceedingly fragile, so that it could not be touched by the fingers without breaking. It is therefore transferred to the crucible, together with a portion of the water contained in the flask. By heating it in the way described, it acquires an amount of cohesion which admits of its being handled without danger of breaking, and at the same time it assumes a distinctly metallic aspect; its volume is also considerably diminished.

In establishments in which large numbers of assays of gold bullion are made, the use of glass flasks for the attack of the cornets by nitric acid is now frequently dispensed with, and an apparatus made of platinum is employed instead. This consists of a shallow dish, furnished with a rack, in which are inserted a number of small vessels, also of platinum, having nearly the shape of an ordinary thimble, with apertures which admit of the free entrance of the acid contained in the dish in which they are enclosed. Into these the cornets are inserted, and nitric acid of the required strength is poured into the outer dish, which is heated, either over a sand-bath or gas-burner. As this acid serves for the attack of all the cornets inclosed in the several small platinum cullenders, a considerable economy of time is effected, since by one operation the acid is charged on the whole of them. While the attack is being made the apparatus is covered by a funnel-shaped dome of glass, communicating, by means of a glass tube, with a condenser in connection with the chimney of the laboratory. In this way a considerable amount of the acid is recovered, and all unpleasant fumes are avoided.

However carefully and skilfully the parting of the auriferous button may be conducted, the cornet of gold will sometimes be found to retain minute traces of silver, by which its weight will be, to a small extent, increased. This increase of weight or *surcharge* is most apparent in cornets obtained from bars containing only a very small quantity of copper, since when the proportion of that metal is considerable, and a large amount of lead has consequently to be employed during its cupellation, a loss of gold takes place by absorption into the cupel. In the assay of bars of moderately cupriferous bullion, the loss of gold during cupellation, and the surcharge of silver remaining after parting, not unfrequently counterbalance one another, and the true fineness of the alloy is directly obtained with a sufficient degree of accuracy. It is, however, necessary, in order to ascertain the amount of surcharge, to have check assays or *proofs* made of pure gold and copper, according to the supposed quality of the alloys.

The following results of numerous experiments made in the Paris Mint upon mixtures of fine gold and copper in the proportions indicated in the Table, afford data for calculating the necessary correction:—

| Copper. | Gold. | Result obtained. | Difference. |
|---|---|---|---|
| 100 | 900 | 900·25 | +0·25 |
| 200 | 800 | 800·50 | +0·50 |
| 300 | 700 | 700·00 | 0·00 |
| 400 | 600 | 600·00 | 0·00 |
| 500 | 500 | 499·50 | −0·50 |
| 600 | 400 | 399·50 | |
| 700 | 300 | 299·50 | |
| 800 | 200 | 199·50 | |
| 900 | 100 | 99·50 | |

The residual silver may be almost entirely removed from the cornet by heating it with fused acid sulphate of potassium in a small clay or porcelain crucible, and extracting the silver sulphate with water containing a little sulphuric acid. The cornet is subsequently dried and ignited. Absolutely pure gold cannot, however, be obtained by this method.

The gold fineness of bullion may be reported either in carats or in thousandths; 24 carats or 1,000 fine being equivalent expressions for absolutely pure gold, 18 carats for 750 fine, and so on; one carat being equivalent to 41·66 thousandths. The carat is subdivided into 4 carat-grains, and each of these again into eighths or thirty-secondths of a carat, giving an ultimate division into 768 parts. Formerly the return was made to one-eighth of a carat better or worse than British standard gold of 22 carats or 916·6 fine; but now the millesimal fineness is generally given to thousandths and tenths of thousandths.

In America it is customary to report the assays of gold ores in money value per ton, at the rate of $20·67 per ounce troy of gold of 1,000 fine, and the same practice is followed with bullion the value in currency being stamped on the bar. The standard values adoped in the principal gold using countries are as follows:—

| Countries. | 1000 (24 Ct.) | 916·6 (22 Ct.) | 900 (21·6 Ct.) | 750 (18 Ct.) |
|---|---|---|---|---|
| United Kingdom, troy oz. | £4, 4s. 10d. | £3, 17s. 10d. | £3, 16s. 6d. | £3, 3s. 9d. |
| United States, „ . | $20·67 | $18·95 | $18·60 | $15·50 |
| France, kilogramme . | Fr.3,444·44 | Fr.3,157·40 | Fr.3,100 | Fr.2,583·33 |
| Germany, „ . . | Mk.2,790 | Mk.2,474·16 | Mk.2,511 | Mk.2,092·5 |

The principal standards in use for coinage are:—

| | |
|---|---|
| Austria and Holland (old ducat gold) . . . . . | 986 and 983 |
| United Kingdom, India, Russia (old), Portugal, Brazil, Turkey . | 916·6 |
| France, Germany, United States, Italy, Spain, Russia (new) . . | 900 |
| Egypt, Mexico, Spanish Philippine Islands . . . . . . | 875 |

Most of the remaining countries have adopted the standard of 900 fine.

In the United Kingdom the only standard for gold plate and jewellery recognised by the Goldsmith's Company for "hall marking" is 18 carats (750) fine; but the lower finenesses of 14, 12, and 9 carats are extensively used for cheaper articles.

In France the three plate standards of 920 ($22\frac{3}{32}$ ct.), 820 ($20\frac{5}{32}$ ct.), and 750 (18 ct.) are legalized, the use of 583 (14 ct.) being only permitted for the manufacture of watch cases intended for exportation.

*Determination by the Touchstone, &c.*—The method of assay already described, although perfectly adapted for the determination of the value of bullion and other unmanufactured products, cannot be conveniently applied to the examination of jewellery, since it would be necessary to destroy the object in order to ascertain its composition. A method is, therefore, employed by which the standard is readily determined to within 1 per cent. of the truth, while the most delicately-chased article is in no way disfigured by the trial. This process essentially consists in rubbing some convenient part of the object to be examined on a hard siliceous stone of a black colour, on which it thus leaves distinct metallic traces. From the colour of these marks, and from their behaviour when treated with nitric acid or a weak solution of aqua regia, the assayer judges of the purity of the gold subjected to examination. The material employed for this purpose, generally known by the name of *touchstone*, is a fine-grained, dark-coloured variety of quartzite, said to have been anciently brought from Lydia, although stones of equally good quality are obtained in Saxony, Bohemia, and numerous other localities.

In order to be enabled to judge of the value of an alloy from the nature of the mark left by it on the surface of the stone, the assayer is furnished with a series of small bars, or *touch-needles*, formed of alloys of copper and gold, of which the composition has been accurately determined. For gold of low fineness which contains some silver alloy, another series of touch needles alloyed wholly or in part with silver, instead of with copper alone, is sometimes used.

The trace left on the stone by the alloy to be examined is successively compared, both before and after the action of an acid, with the different marks obtained from these several needles, and it is supposed to possess a similar composition to that of the needle whose mark agrees most closely with it under both these circumstances. The acid most commonly employed for this purpose is nitric acid of sp. gr. 1·28, to which about 2 per cent. of hydrochloric acid is sometimes added. In making these assays, the first streak obtained on the stone cannot be employed to ascertain the composition of the object examined, as the surface of jewellery is rendered, by the process of "colouring," of a higher standard than that of the alloy of which it is composed throughout. For this reason the object must be passed once or twice over the surface of the stone, in order to remove the superficial coating of richer alloy, before making the streak, from the comparison of which with those of the needles the commercial value of the mixture is to be determined. This method, although affording much less accurate results than those obtained by inquartation and parting, is nevertheless for some purposes sufficiently exact.

The *colouring*, as it is called, of jewellery, is effected by externally dissolving out the copper with which it is alloyed, and thereby exposing a superficial facing of fine gold. To produce this effect, the object to be coloured is first heated in a gas jet or spirit lamp, and then plunged into a weak solution of nitric acid, by which the copper on its surface is removed. The same effect is also produced by placing, for a few minutes, the object to be coloured in a paste composed of a mixture of alum, common salt, and saltpetre.

## MECHANICAL AND METALLURGICAL TREATMENT OF GOLD.

The mining and metallurgy of gold are so intimately connected that it would be difficult to treat intelligibly of the one without also giving some description of the other. From the great difference existing between the density of gold and that of siliceous gravels, it is easily separated from them by washing and amalgamation. The methods employed for this purpose, however, vary not only with the localities in which the operation is carried on, but also in accordance with the nature of the minerals with which the gold is associated.

In alluvial or placer mines the metal is found imbedded in strata of clay, sand, and gravel; while in vein mines it forms one of the constituents of mineral veins or lodes. In placer mining the auriferous earthy material, usually called "pay-dirt," is exposed to the action of water, by which the clay is carried off in suspension, and the sand and gravel removed by the force of the current. The gold, on account of its high specific gravity, either remains behind in the apparatus employed, or is caught and amalgamated with mercury.

In quartz mining the gold-bearing veinstone is ground to a fine powder, and the gold is either caught on the rough surface of blankets or skins, over which the finely-divided material is borne by a stream of water, or it is amalgamated by bringing it into contact with metallic mercury or amalgamated copper plates.

### Placer Mining.

Water is the principal agent employed by the placer miner, and its abundance or deficiency is to a great extent the measure of the work to be performed and of the profits to be realized. Placer mines may be divided into two classes, deep and shallow. In the former the pay-dirt lies at considerable depths, whilst in the latter it is found near the surface. Many deep diggings are worked on what have evidently been ancient river-beds, and are sometimes covered by flows of lava. Shallow diggings, which are found in gullies and ravines, as well as in the beds of rivers, frequently owe their richness to the redistribution, by modern streams, of the gravels of these ancient rivers.

The appliances made use of by the placer miner are exceedingly simple, but at the same time singularly ingenious and effective.

**Pan.**—This is the simplest of all contrivances for washing auriferous materials. It is used in all branches of gold mining, either for washing, or as a receptacle for gold, amalgam, or for rich dirt. The pan is made either of stiff tin-plate or of sheet-iron, with a flat bottom about 12 inches in diameter, and has sides from 5 to 6 inches in height, sloping outwards at an angle of 45°. Sheet-iron is to be preferred to tin-plate, because it is stronger and does not amalgamate with mercury. The process of washing is conducted in the following way:—After being about three-fourths filled with dirt, the pan is placed in water, which should not be more than a foot in depth, so that it may rest on the bottom, while the miner inserts his fingers in and under the mass, in order to lift and stir it, in such a way that it may become thoroughly wetted throughout. The pan is then held by the two sides, that portion of it which is towards the body being raised, and the opposite edge lowered. He now commences shaking it from side to side, taking care that the whole of the dirt is under water, and that a little of it can escape over the outer edge. Assisted by the shaking, and by the rolling of the gravel from side to side, the clayey portion of the mixture rapidly becomes suspended in water and forms a thin mud which escapes over the side, whilst clean water is continually flowing into the pan. The light sand follows the mud, while the larger stones and lumps of tough clay remain. The stones and pebbles collect on the top of clay, and are scraped together with the fingers, and thrown out. This process is continued, care being taken to gradually lower the outer edge of the pan until all the clayey matter has been swept away by the water, and until gold and a certain amount of magnetic iron-sand alone remain.

To get rid of this black sand, the contents of the pan are dried, and a small quantity is placed in a "blower," consisting of a shallow tin scoop open at one end. The miner holds this with the open end from him, and gently blows out the sand, leaving the particles of gold behind. During this operation the blower must be occasionally shaken, so as to bring all the particles of black sand within range of the current of air.

The pan is also constantly employed for separating amalgam from sand or pyrites, and for cleaning up rich dirt collected in the cradle, long tom, or sluice.

CRADLE.—The cradle, or rocker, is, after the pan, the cheapest and most simple apparatus employed for gold-washing. It rests on two rockers, and its general appearance is not unlike that of a child's wooden cradle. The cradle box is 40 inches in length, 20 in breadth, and is at one end from 18 inches to 2 feet in depth, whilst at the other it is sloped off to about 4 inches only. On the deeper end of the cradle stands a hopper or riddle-box, 20 inches square, with sides from 4 to 6 inches high. The bottom of the riddle is of sheet-iron, perforated with holes half an inch in diameter, and the box itself is so constructed that it can be either slipped into its place or lifted off without difficulty. Under the riddle is an apron of wood or cloth, attached to the sides of the cradle and sloping towards the upper end of the arrangement. Across the bottom of the cradle-box are nailed two wooden bars, each about an inch square, known as *riffle-bars*, one near the riddle and the other at the shallow end.

The dirt to be washed is shovelled into the hopper, and the cradler sits beside his machine; with one hand he pours water, dipped with a ladle from a pool at his side, upon the dirt, and with the other he imparts to it a rocking motion. By means of the water, aided by the rocking, the dirt is disintegrated and carried through the riddle, falling on the apron, by which it is carried to the head of the box, whence, as the bottom has an inclination towards the shallow end, it runs downwards and escapes, leaving the gold, black sand, and heavier particles of gravel behind the riffle-bars.

The pay-dirt contains many large stones; and such as give a too considerable shock to the cradle, by rolling from side to side, are picked out by hand, and, after being examined to see that no particles of gold are adhering to them, are thrown away. All the smaller ones are allowed to remain until a hopperful has been washed, so that nothing but clean stones remain, and then the cradler, rising from his seat, removes his hopper, and, with a jerk, throws out its contents. The whole process of washing with this arrangement is merely a repetition of the manipulations described. The hopper is about one-third filled with pay-dirt, and water is poured in from a ladle held in one hand, whilst with the other the cradle is rocked. The cleaning-up is done by removing the hopper, taking out the apron, scraping all the dirt from the bottom of the cradle with an iron spoon, putting it into a pan, and washing off the impurities as already described. Mercury is sometimes, but not generally, used in the cradle.

TOM.—The tom was formerly much used by Californian miners, but is now seldom employed, having become generally superseded by the sluice. It consists of a wooden trough about 12 feet in length, 18 inches in width at its upper end, and gradually widening to 30 inches at the lower. Its sides are 8 or 9 inches high, and at the lower end its bottom is of sheet-iron perforated with holes half an inch in diameter. This sheet-iron is turned up, so that the water cannot flow over it, but passes down through the perforated riddle into a riffle-box furnished with transverse bars.

The tom itself is arranged at a considerable angle, and a stream of water is admitted at the higher end. The pay-dirt is thrown in at the head and kept constantly stirred with a shovel, care being taken to throw back to the upper part of the trough such pieces of clay as are not sufficiently disintegrated.

The tom can be most advantageously employed where the amount of stuff to be washed is not large and the gold is coarse. The riffle-box is charged with mercury, and, as its contents are constantly kept in motion by the falling into it of the water from the riddle above, a considerable proportion of the gold is caught, although there is always a notable loss of the finer particles.

SLUICE.—The sluice is now the great washing apparatus of California, and washes nearly all the pay-dirt and produces the greater portion of the placer gold of that country. It is generally a long wooden trough, through which a stream of water constantly flows, and into which the auriferous material is shovelled. Its length is always several hundred feet, and sluices more than a thousand feet are not unfrequently employed.

The width is often about 18 inches, but is sometimes as much as 6 feet. It is made of sawn pine-planks, 1½ inch thick, in sections or boxes, from 12 to 14 feet in length, the bottom plank being often sawn 4 inches wider at one end than at the other. By this means the narrow end of one box is made to fit into the broad end of the next, so that the sluice is composed of a long succession of boxes fitting one another by spigot and faucet joints, but not otherwise fastened. These boxes stand on trestles, and have a slope or "grade" varying from 8 to 18 inches in 12 feet. The amount of inclination given to them is necessarily varied in accordance with the nature of the stuff to be washed. The dirt often contains numerous large stones and boulders, all of which must be carried off through the sluice by the action of the water rushing down its channel. Sometimes a grating or "grizzly" is used to keep back the stones, which are removed at intervals by men with four-pronged steel forks. When much clay is present the sluice should have a considerable grade, and as a rapid current is more liable to carry off fine particles of gold than a slower one, the length of the arrangement should be proportionately extended. Economy and facility of working require that the sluice should not be much above the surface of the ground, and the inclination is therefore, to a certain extent, modified in order to conform to local circumstances. Sometimes the upper portion of a sluice has a steep grade for the purpose of more readily disintegrating the dirt, whilst the lower end has a less inclination given to it with the view of more effectually collecting the gold. The clay of ordinary pay-dirt is completely disintegrated in the first 200 feet of a sluice with a low grade, and its prolongation beyond that point is only of use for collecting the liberated gold. In certain cases, however, the clay met with is so exceedingly tenacious that it will roll in large balls through a quarter of a mile of a steep grade sluice with a large head of water and be scarcely diminished in size.

The bottom of the sluice is provided with riffle-bars for the purpose of retaining the gold, which would readily pass off from the surface of the boards, and these would be rapidly worn out, unless thus protected. Most commonly the false bottom is composed of longitudinal riffle-bars, from 2 to 4 inches in thickness, from 3 to 4 inches wide, and about 5 feet 6 inches in length. Two sets of bars are fitted into each box, and are wedged in, from 1 to 2 inches apart, with a transverse bar of the same width and thickness, placed between each set of riffles. The bottom of the sluice is therefore divided into parallelograms about 5 feet 6 inches in length, from 1 to 2 inches wide, and from 2 to 4 inches in depth. In these spaces the gold, amalgam, and quicksilver are retained.

Instead of riffle-bars sawn longitudinally with the grain of the wood, "block-riffles," cut across the tree and standing in the sluice with the grain upwards, are often employed. These are found to be much more durable than those of the ordinary kind, but require a different arrangement in the boxes. In some sluices, and particularly those employed for hydraulic mining, the block-riffles are placed transversely in the box, and kept at a distance of about 2 inches apart by strips of wood interposed between them at the bottom, but of less depth than the blocks themselves.

In small sluices the riffles are sometimes placed in zigzag order on the bottom of the boxes, but not touching the side at one of their extremities. These are set at an angle of 45° with the axis of the sluice, and just below the open space left between the first riffle and the side of the box, another bar starts at right angles to the first, and an open space is again left at the lower end of this bar. This is continued down to within a short distance from the end of the sluice, where ordinary riffle-bars are inserted. In sluices thus constructed, much of the water and light mud flows directly over the riffle-bars, whilst the heavier materials, falling to the bottom, assume a zigzag course. A vessel containing mercury placed near the head of the sluice allows it to fall drop by drop into the trough, and this, following the course of the riffle-bars, overtakes the gold, and amalgamates with it. These zigzag riffles are retained in their places by being nailed to the bottom of the box.

The height of the sides of sluice-boxes varies from 9 inches to 2 feet, and the stream of water employed has never a less depth than 2 inches over the bottom. In most instances the sluice extends throughout the length of the claim in which it is situated, and the auriferous dirt is thrown in with shovels, which requires from four to twenty men constantly at work.

In nearly all sluices mercury that has been before used for the same purpose, which is considered better for gold-catching than that fresh from the flask, is put in above the riffle-bars, at various places along the boxes. Oil and grease tend to prevent amalgamation, and must be carefully avoided.

The usual method of arresting very fine gold is to cover the surface of a copper plate with quicksilver, and to allow the dirt and water to flow slowly over it, with a depth of not more than a quarter of an inch. The amalgamation of a copper plate is effected by adding a few drops of nitric acid to water, covering a little mercury in the bottom of a saucer, and then rubbing the liquid with a rag over the surface of the metal. Decomposition of nitrate of mercury quickly covers the surface of the copper with a bright coating, to which metallic mercury readily adheres.

A plate which has been once thus amalgamated does not again require similar treatment, but a little additional quicksilver must from time to time be sprinkled upon it, as the gold gradually collects and forms a solid amalgam. The plate employed, which is often about 3 feet wide, and 6 feet in length, is set nearly level. In very large sluices the stream is so divided as to flow over several distinct plates, in order to secure a slow and shallow current. It is evident that with a rapid current, or with deep water, particles of light gold would pass off without coming in contact with the surface of the amalgamated plate.

When the surface of a plate has become covered by auriferous amalgam it is supposed to act more efficiently than a new one, and at the time of cleaning up it is sometimes coated with a hard brittle mass to a thickness of $\frac{1}{4}$ inch. To remove this the plate is warmed until the hand cannot long remain in contact with it, by which treatment the amalgam becomes softened, and is then readily scraped off. The plate, after being sprinkled with mercury, is again ready for use. The mixture of mud and water is admitted to the copper plate through a riddle made by piercing a thin iron plate with holes about $\frac{1}{10}$ inch in diameter. This is often placed above the copper plate, and prevents its surface from being swept by the gravel and coarser materials which pass along it.

Shortly after the water and dirt have begun to pass through the sluice all the spaces between the different riffle-bars become filled with sand and gravel, which is, however, in a constant state of agitation. The coarser grains are arrested near the head of the sluice, while the finer particles are carried to more considerable distances. In sluices where the dirt operated on contains much coarse gold, the mercury is often introduced from 40 to 60 yards below the head, as the coarse metal, by virtue of its greater density, becomes readily separated from the earthy materials with which it is mixed.

The separation of the gold, amalgam, and mercury from the sand and gravel in the bottom of the sluice is called "cleaning up," and the period which elapses between one cleaning-up and the following is called a "run." A run ordinarily extends over eight or ten days, but in most instances the work is only carried on during daylight, although in some cases it is continued day and night. When the period fixed on for cleaning up arrives the throwing-in of dirt ceases, and the water is allowed to run until it becomes perfectly free from turbidity. Five or six sets of riffle-bars at the head of the sluice are now taken out, and the dirt, which has accumulated between them, is washed away, while the gold and amalgam are arrested by the first of the remaining set of riffles, whence they are removed by the aid of a spoon or scoop, and placed in a washing pan. More riffle-bars are now removed, and the gold and amalgam are again collected. This is repeated until all has been taken out, and the whole of the gold and amalgam is transferred to the iron pan.

The quicksilver and amalgam taken from the sluice are put into a buckskin or piece of canvas, and pressed in such a way that the liquid metal passes through, while the amalgam is retained. This amalgam, from which the mercury has been carefully pressed out, contains about one-third its weight of gold. The amalgam is then heated to drive off the mercury, and the gold which remains is in the form of a spongy mass of a light yellow colour.

The removal of the mercury may be effected either in a close retort or in an open iron pan. In the first case the quicksilver is recovered for subsequent use, whilst in the second, it is volatilized and lost. The pan is, however, often employed by placer miners.

Large sluices are sometimes paved with stones, which make a more durable bottom than wood, and one that catches fine gold almost as effectually. On the other hand, cleaning up is more difficult, as is likewise the re-laying of the bottom afterwards. The stones used are water-worn pebbles, of a somewhat flattened form, of which the greatest diameter is from 6 to 8 inches.

The ordinary sluice, as already described, mainly consists of a series of wooden boxes, but, in some cases, these are dispensed with, and the arrangement is then called a "ground-sluice." This contrivance is employed for washing dirt in localities where water is only abundant for a few weeks after heavy rains, and where, consequently, it would not pay to erect wooden sluices.

To prepare a ground-sluice a stream is first directed through a small channel, which the miners, aided by the current, endeavour constantly to enlarge, and when it has become sufficiently deep they detach, by means of crow-bars, the high banks, so that the pay-dirt falls into the ditch, where it becomes rapidly disintegrated. A large pebble should be introduced at intervals into the bed of the sluice, for the purpose of arresting the gold, since, if the bottom were smooth, and without irregularities, the precious metal would be carried away by the current. No mercury is employed in the ground sluice, but the concentrated dirt collected is finally cleaned up in a wooden sluice, or tom.

HYDRAULIC MINING.—Hydraulic mining is resorted to only in situations where the pay-dirt is of considerable thickness and where water is abundant. It may be regarded as the highest branch of placer mining, since by it a larger amount of dirt is washed in a given time, and at a less expense, than by any other process. Hydraulic claims are necessarily situated in hilly districts, as it is not only necessary to be provided with a column of water of considerable height, but also to find in the valleys below the sluices a receptacle for the enormous amount of *débris* resulting from the operation.[1] Whatever may be the depth of the auriferous deposit, the whole of it should, if possible, be removed to the bed-rock. This is effected by the action of water issuing at a high pressure from metallic nozzles, and directed against the more or less indurated alluviums operated on. This disintegration of the auriferous material goes on simultaneously with the washing of the resulting gravel, and is effected by the same supply of water.

In California, the water employed in hydraulic claims is generally purchased from one of the large public companies formed for supplying the gold diggings with this essential requirement of the miner. Its cost is from 10 c. to 20 c. per miner's inch per working day, and the consumption of each mining claim, worked on a tolerably extensive scale, may be taken at about 300 miner's inches. A miner's inch is the quantity of water which will flow during ten hours through an aperture 1 inch square, under a mean head of 6 inches; and 300 miner's inches are equal to 284,210 cubic feet, or about 1,772,000 imperial gallons. Under ordinary circumstances, from 3,000 to 3,500 cubic yards of gravel and soft conglomerate may be removed and washed daily by this expenditure of water.

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[1] So much of the low-lying land in some parts of California has been buried under sands and gravels from hydraulic mines, that the legislature of the State has recently interfered, on behalf of the agricultural interest, in such a way as to considerably restrict this branch of mining industry.

The installation of a hydraulic washing is commenced by bringing in a stream, by means of a "flume" or aqueduct, to the head of the mining ground at a height which may be from 150 to 400 feet above the level of the bed-rock, where it is conducted through a large wooden tank, into which it constantly flows. This is provided with a suitable valve, and from it the water is most commonly conveyed to the bottom of the claim through wrought-iron pipes. When a great head of water is employed the lower pipes, which have to withstand a very great pressure, are made of double-rivetted boiler plates, and in connection with these are the several discharge nozzles. Some of these are as much as 5 inches in diameter, mounted on ball and socket joints, so balanced as to be readily turned in any direction, either by a lever or by racks and worm pinions. The amount of manual labour necessary for carrying on the operations of a hydraulic claim, is exceedingly small in proportion to the amount of work done, since, in addition to the men engaged in directing the various nozzles, only one person is usually employed in attending to the sluice, so as to remove obstructions, and prevent its becoming choked by the dirt and boulders washed from the face of the slope. In order to render evident the enormous advantages possessed by this over every other system of placer mining, it may be stated it has been estimated that, taking a miner's wage at $4 per day, the cost of treating a cubic yard of gravel by the various processes which have been described will be approximately as follows:—

| | |
|---|---|
| By the pan | $20·00 |
| „ „ cradle | 5·00 |
| „ „ long tom | 1·00 |
| „ „ sluice | 0·33 |
| „ „ hydraulic process | 0·05 |

The quantity of dirt, however, that can be washed by a hydraulic jet in a given time, depends on various circumstances, such as the supply of water, the height of the column, the tenacity of the material, and the amount of moisture it may happen to contain. More work can usually be done in winter than in summer, since, from the greater dampness of the stuff during that season, it becomes more easily disintegrated. In some hydraulic claims, in which the pay-dirt is cemented into a kind of conglomerate, blasting is resorted to as a means of facilitating its removal by the action of water. For this purpose a tunnel is driven along the upper surface of the bed-rock into the hill, which may be 150 feet in height, and a number of kegs of powder (frequently above a hundred) are introduced. The tunnel is now re-filled with earth, and the powder is exploded by the use of a properly-arranged slow-burning fuse. The explosion, which often makes comparatively little noise, loosens and shatters thousands of cubic yards of the surrounding hill, and materially facilitates its subsequent removal by the water thrown against it through the nozzles.

In hydraulic claims, generally, all the alluvium is removed to the bed-rock, but in some cases working has to be suspended long before this point is reached, from the circumstance of the conformation of the country not allowing of an outlet for the water at that depth. The cheapness and expedition of this process admit of very poor alluviums being treated with advantage; and in some exceptional cases, claims in which the dirt afforded gold to the value of only 3 c. per ton of 15 cubic feet (2¾d. per cubic yard), have been worked with satisfactory results.

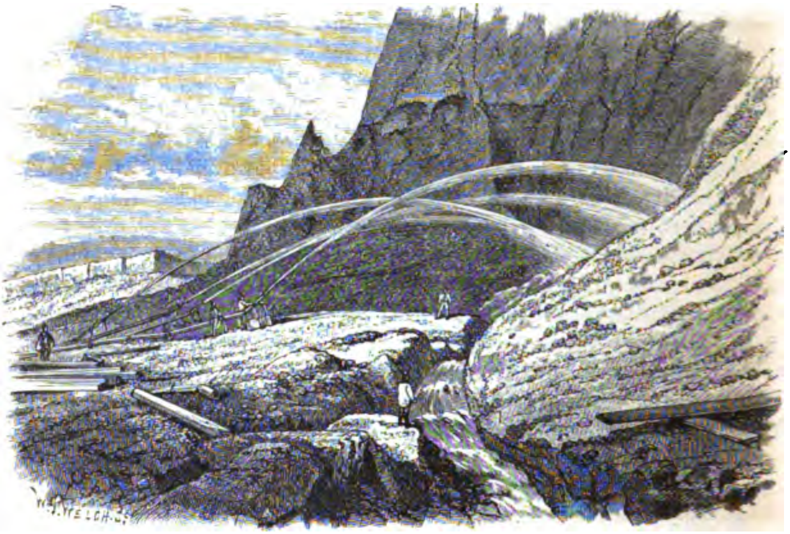

Fig. 221.—Hydraulic Mining; Timbuctoo, California.

The accompanying woodcut (fig. 221), from a photograph of the Palm claim, Timbuctoo, Yuba county, California (1865), will afford a good idea of the general appearance of an extensive hydraulic washing.

In many claims a larger amount of water is required for piping down the bank than for washing the dirt removed, and in such cases the sluice cannot be kept properly supplied without the aid of gunpowder.

Some idea of the magnitude and importance of the water interests in the United States may be gathered from the following statistics:—

| | |
|---|---|
| Total length of ditch lines . . . . | 10,783 miles. |
| Total length of ditches proper . . . | 10,183 ,, |
| Total length of flumes . . . . . | 450 ,, |
| Total length of pipes . . . . . | 150 ,, |
| Maximum capacity of ditch lines, gallons per 24 hours . . . . . . . | 7,560,000,000 |
| Total cost of plant, including reservoirs, &c., and excluding cost of water right . | $27,056,942, 11c. |

## EXTRACTION OF GOLD FROM AURIFEROUS VEINSTONE.

The methods employed for the extraction of gold quartz from the mines differ in no respect from ordinary mining operations applied to the systematic working of mineral veins. After the quartz has been obtained by the operations of mining, it is necessary that it should be reduced to a state of fine division before the separation of the gold it contains can be effected. Various contrivances are employed for this purpose, and one of the simplest and most primitive is the arrastra.

ARRASTRA.—This apparatus, as constructed for the treatment of auriferous ores, is essentially similar to that used in Mexico. (See fig. 204, p. 685.)

The charge of such an arrastra is about 4 cwts. of quartz, previously broken to the size of beans, and requires to be ground during from four to five hours, to reduce it to a sufficiently fine state of division. Water is now added and thoroughly incorporated with the ground ore, care being taken that the resulting mud be not too liquid. When the paste has assumed the proper consistency, that of thick cream, mercury is scattered over its surface, by squeezing it through a piece of canvas, in the proportion of about 2 ounces to every ounce of gold supposed to be contained in the quartz. The grinding is now continued for a further period of two hours, by which the mercury becomes divided into minute globules, which are disseminated throughout the mass, and effect the amalgamation of the gold. When the operation is supposed to be complete, more water is let in on the surface of the paste, and the mullers again set slowly in motion. By this treatment the lighter earthy particles become suspended in water, while the heavier amalgam gradually collects at the bottom. This result is supposed to have been attained at the end of about half an hour, when the mud is run off, leaving the gold amalgam and mercury at the bottom. Another charge is now introduced, and the process is repeated as before.

The run with a rude arrastra of this description generally extends over a week, but sometimes over a considerably longer period. For the purpose of cleaning up, the stones of the bed require to be taken up, in order to collect the amalgam which settles between them, and the whole of the mud must be removed and carefully washed.

The amount of work performed by this machine is small in comparison with the power expended; but the proportion of gold extracted is generally larger than is obtained with more expeditious and more complicated apparatus. The arrastra is, therefore, not unfrequently employed as a means of making a practical trial of the value of gold quartz, before proceeding to erect expensive machinery for its treatment on a large scale. In California this extremely primitive contrivance for working auriferous quartz has now almost entirely disappeared. Those, however, who travelled through that country about the year 1854 will remember frequently falling in with families of Mexicans, who, in some secluded valley, with a couple of wretched mules harnessed to a rude arrastra, managed to pick up a scanty living from the outcrop of some quartz vein.

CRUSHING-ROLLS.—Many years since, crushing-mills were introduced at the Pestarena Mines in the North of Italy for the reduction of auriferous ore to a fine state for amalgamation, and these still afford satisfactory results. Latterly crushing-rolls have been applied by Krom of New York to a similar purpose. The shells are of steel about 2¾ inches thick, from 26 to 30 inches diameter, and from 15 to 16 inches in length on the face. The pillow-blocks of one of the rolls are firmly bolted to a bed-plate, while the bearings of the other are on the end of a swinging arm which can be brought near to or more distant from the face of the fixed roll. Each roll is driven by an independent belt at a speed of from eighty to one hundred revolutions per minute. The rolls are "housed," or covered in, and an exhauster is employed for the purpose of withdrawing the dust as soon as produced.

By means of a stone-breaker, a coarse- and a fine-sand crushing-mill, quartz is reduced sufficiently fine either for pan or plate amalgamation.

One pair of steel rolls at the Bertrand Mill, Nevada, is stated to have reduced 20,000 tons of quartz to a sufficiently fine state for milling.

PULVERIZERS.—Of late years numerous attempts have been made both in Europe and in America to substitute pulverizers for the stamping-mill, but all such attempts have ended in practical failure. The drawbacks usually attendant on this class of apparatus are an excessively high speed, requiring for its maintenance a large expenditure of power, inducing costly wear and tear, or a slow speed, and great friction between the heavy grinding parts.

GRIZZLIES AND STONE-BREAKERS.—The quartz as it comes from the mine, large and small pieces together, is usually tipped upon a grizzly or screen inclined at an angle of 42°, the bars of which are set 2½ inches from each other. The small stuff, which passes through the openings between the bars, falls into the hopper of the stamps, while the coarser lumps are reduced to sizes from and below 1½ inch cube, and also falls into the same receptacle. The dimensions of a grizzly will vary with the size of the stone-breaker and weight of quartz to be treated per hour. When the jaw of the stone-breaker measures 15 inches by 9, the grizzly may be 4½ feet wide and 12 feet long, formed of twenty-four bars of wrought-iron 1 inch thick and 2 inches deep.

Stamping-Mill.—Probably nineteen-twentieths of the quartz crushed for the purpose of extracting its gold is pulverized in the stamping-mill. In fact, this may be said to be the only machine extensively employed for the reduction of auriferous veinstone. It essentially consists of a series of heavy pestles enclosed in a rectangular mortar, each of which is successively lifted by means of a cam, and then allowed to fall with its full weight on the ore operated upon. A constant supply of mineral is kept up in the mortar, while that which has become sufficiently reduced in size is gradually removed, by suspension in water, through the apertures of properly arranged sieves or screens. In some cases the stems or lifters are of wood, as in the old German and Cornish stamping-mills, but the modern machine, figs. 222, 223 (pp. 776, 777), is more frequently employed. The size of the apertures in the screens is varied in accordance with the dimensions of the particles of gold in the rock under treatment; but it is evident that with very small apertures the amount of rock crushed, all other conditions being equal, will be less than when a coarser grating is employed. Screens are commonly made of thin sheet-iron, in which are punched, at regular intervals, holes of the diameter of a large sewing needle. The auriferous material having become reduced to the state of finely-divided sand, it becomes necessary to find means for the concentration and separation of the released gold. This may be effected either with or without mercury.

The depth of the surfaces of the dies below the bottom of the screens is a point upon which there is among mill men much diversity of opinion. The practice in America and Australia is to let the top of the dies stand from 1 to 3 inches below the bottom of the screen. At St. John d'el Rey, where the gold, in an extremely fine state of division, is associated with iron pyrites, "deep stamping" is alleged to give more favourable results than shallow stamping. At that mine the surface of the dies is about 15 inches below the bottom of the screen. The weight of the modern round stamp head varies from 500 to 900 lbs. In dry crushing-mills the drop of a head 850 lbs. weight is from 7 to 8 inches. In wet crushing-mills the length of drop is perhaps rather less, from 5 to 7 inches, especially when the rock is soft and the gold fine. The number of drops per minute ranges from 80 to 110, the average for an ordinary mill being about 90. When the die is new its face is almost on a level with the top of the discharge opening. Amalgamation in California is now generally effected, both inside and outside of the mortar, by means of amalgamated copper or electro-silvered plates against and over which the pulp either dashes or flows. By mortar and plate amalgamation from 50 to 80 per cent. of the free gold in the pulp is obtained. "Low-feeding," that is, supplying a small quantity of rock to the battery as often as required, is carefully observed in all well-ordered mills, as the blow of the stamper is held to be more effective on a thin than on a thick stratum of stone. Some diversity of opinion prevails among mill men as to the order in which the several heads should be lifted and dropped in the mortar-box, as well as regarding the number of heads contained in each box. In America the number is invariably five. In Australia also five heads are generally grouped together, except in the more recent pneumatic stamps, which have only a single head. Five heads are constructively very convenient, and this is the number generally contained in each mortar-box; the order in which the stampers are respectively lifted and dropped in various five-stamp batteries is given below:—

| | | | | | |
|---|---|---|---|---|---|
| California | 5 | 3 | 1 | 4 | 2 |
| | 1 | 3 | 5 | 2 | 4 |
| Colorado | 1 | 4 | 2 | 5 | 3 |
| Australia | 3 | 4 | 2 | 1 | 5 |

Automatic feeders are now frequently employed in America for delivering the quartz to the mortar-boxes. One feeder is attached to each box, and placed immediately behind it.

*Amalgamation in Battery.*—When mercury is used in the batteries they are often furnished with amalgamated copper plates of about 5 inches in width extending the whole length of the battery-box or mortar. One of these is placed on the feed side, and the other on the side of the discharge, the former being protected by the iron lining of the feed-hopper, and each having an inclination of about forty-five degrees. When these are not employed, the auriferous amalgam accumulates in the spaces between the dies, as well as between the dies and the sides of the box. The quartz, previously broken to a convenient size, is supplied by the feed aperture. A small stream of water flows into the battery-box through a gas-pipe, and a little mercury is sprinkled into it, by the feeder, at intervals of about an hour, and in quantities varying with the estimated richness of the rock which is being worked. For its collection in the battery an ounce of gold requires about an ounce of mercury, but when the gold is in a finely divided state the addition of a small excess of mercury is advantageous. The proper proportion is, however, readily arrived at by closely watching the discharge. When any particles which pass through the screens are observed to be dry and brittle more mercury must be added. If, on the contrary, soft and pasty globules of mercury pass off, the supply of that metal requires to be diminished. The amalgamation of gold is satisfactorily effected when the proportion has been properly adjusted, excepting in the case where the gold is coated by minerals which interfere with its combination with mercury. When the rock contains coarse gold, and a proper supply of mercury has been introduced, from 60 to 80 per cent. of the precious metal is caught in the battery. When, as is sometimes the case, the gold obtained is in a very finely divided state, and is associated with silver ores or other sulphides, the results are less satisfactory. In some instances, when such ores are under treatment, the alloy obtained, after the removal of mercury by distillation, only yields about one-third of its weight of gold, while the amalgam is spongy and of a dark colour, consisting of an aggregation of minutely divided particles. Amalgam of this description is light, and consequently difficult to collect, whether by riffles, amalgamated copper plates, blankets, or by any of the other appliances employed for the purpose. When, therefore, the rock operated on is of such a nature as to yield an amalgam of this description, amalgamation in the battery is not to be recommended, since this spongy product, being liable to float off, in spite of all the precautions taken to arrest its progress, is even more difficult to catch than finely divided gold. For the purpose of collecting the particles of gold and amalgam which escape through the screens, various contrivances are resorted to. As, however, these differ but little, whether mercury be added in the battery-box or otherwise, it will be sufficient to describe some of the more important.

BLANKETS.—At Grass Valley, one of the most important quartz-mining centres of California, a large proportion of the gold has been obtained from mills, in connection with which blankets have been employed. In such mills the sand and water, escaping through the screens, are conducted over the surface of blankets forming the lining of shallow sluices, inclined at an angle of from three to four degrees with the horizon. Beyond the blankets are amalgamated copper plates, which are again followed by some contrivance for collecting the auriferous materials which have not been arrested in the upper portion of the apparatus. Finally, there is generally a long tail-sluice for collecting any auriferous sulphides which would otherwise be lost. The troughs in which the blankets are placed are from sixteen to eighteen inches in width, with a regular longitudinal slope, care being taken to lay them perfectly level in a transverse direction, so that an equal depth of water may flow over every part of the bottom.

The blankets used for this purpose are generally woven from long wool, and are of such a width that when wetted and fitted closely to the bottom and sides of the trough they extend about half an inch beyond the latter. In laying them in the sluices they are so placed as to overlap one another like the slates on a roof, in order that the water flowing from the upper one may run directly over that next in the series, without allowing any sand to find its way between the bottom of the sluice and the covering of blankets. The troughs are made in two or more lengths, and are so disposed that the sand and water flowing from the first, fall upon the second from a height of three or four inches. This arrangement of batteries and sluices[1] will be understood by reference to the woodcuts, figs. 222, 223, the first of which is a longitudinal section, and the second a plan. In front of the battery-box, A (fig. 222), is a water-tight trough, B, of the same width, which has an opening, *b*, communicating with the sluice, C. There is also a second aperture, *b′*, at the end, which, like the first, can be closed, either by a wooden plug or by wet blanketing. Before each battery is a sluice, C, and between each pair is a third, C′ (fig. 223),

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[1] These are also called blanket-tables and blanket-strakes by English gold-miners.

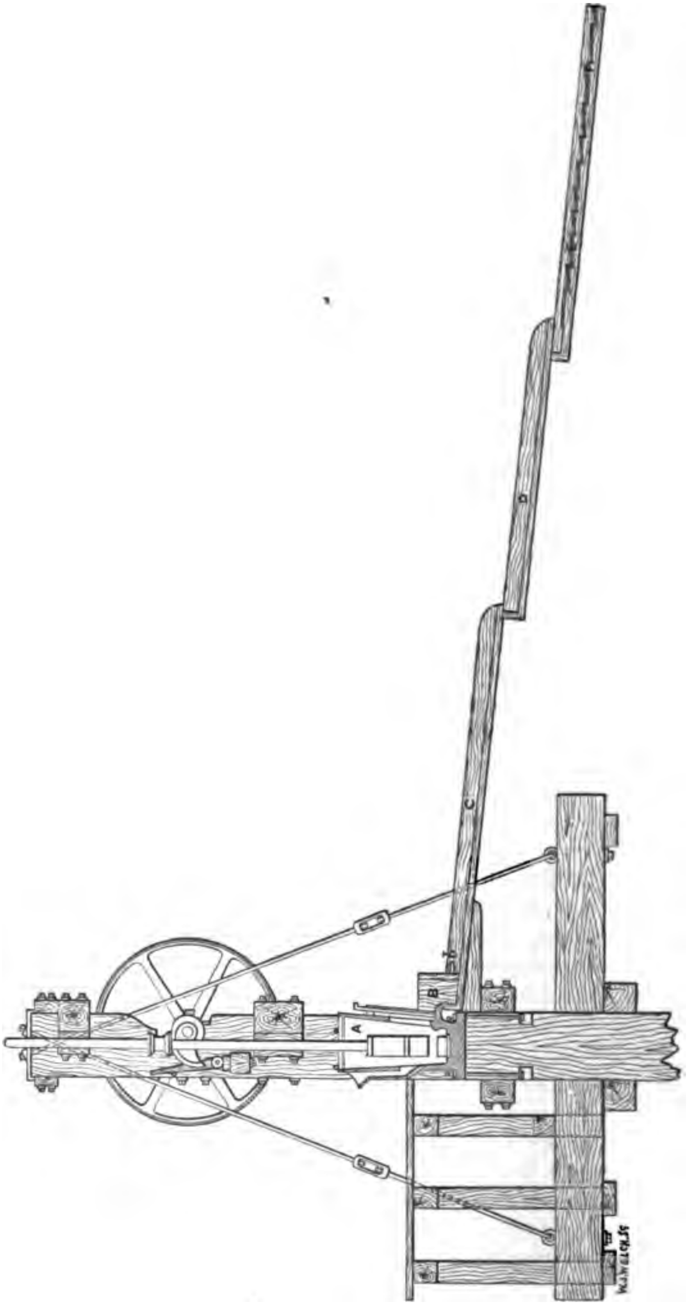


Fig. 222.—Stamping-Mill, with Blanket-Sluices and Riffles; longitudinal section.

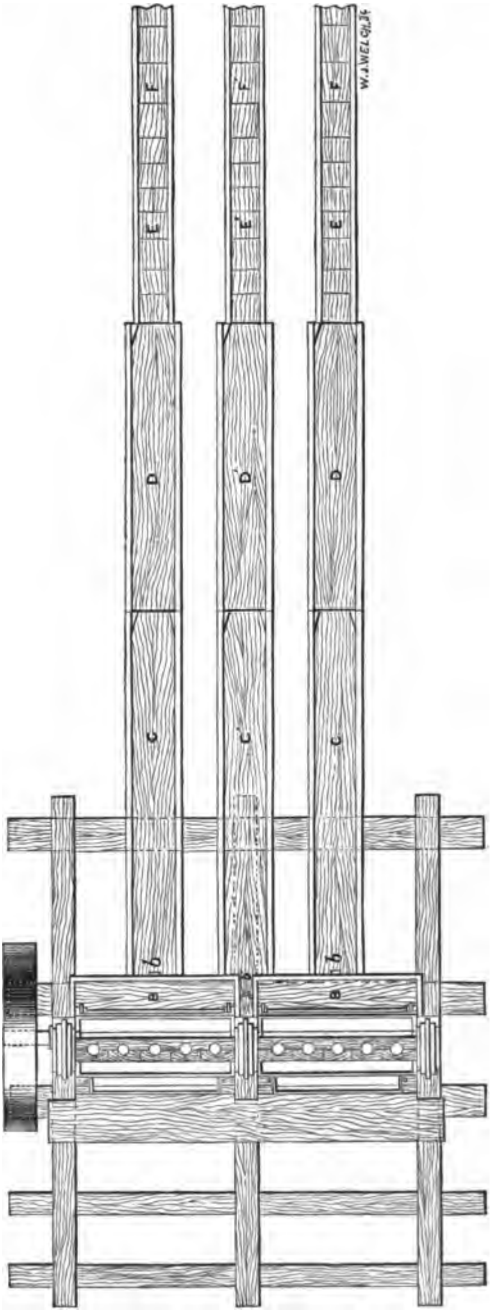


Fig. 223.—Stamping-Mill, with Blanket-Sluices and Riffles; plan.

which is used when either of those on each side of it is thrown out of action to remove the blankets. When in the ordinary course of working, the water from the batteries, carrying the crushed ore in suspension, passes through the troughs, C, and flows over the blankets with which they are lined, a large proportion of the gold and other heavy minerals in the veinstuff becomes entangled in the fibres of the wool, while the lighter particles of quartz are carried off by the current and escape at the lower end of the trough. After a time the blankets become covered with a deposit of the heavier particles in the ore, and the catching action of the fibres ceases, so that to maintain their efficiency they are frequently washed, and subsequently replaced in their respective troughs. At the upper end of the sluices, where the deposit is heaviest, this operation is performed at intervals of fifteen or twenty minutes, in the following manner. The orifice, *b*, communicating with one of the sluices, C, is closed, and the aperture, *b*′, in connection with the central sluice, C′, standing between the two batteries, is opened. The water is thus turned off from the sluice, C, while the discharge from the battery is directed through the central trough, C′. As many of the blankets in the first trough, C, as may require it, are now taken up and folded to prevent the loss of any of the adhering matter. They are then taken to a cistern prepared for that purpose, and, after being carefully washed, are again laid in the trough, from which they were removed. The discharge from the battery is now cut off from the sluice, C′, and again admitted into its original channel; the same operation being repeated whenever it is found necessary to wash up the blankets on the sluices, C. When it is found requisite to remove the blankets from the intermediate trough, C′, it is done during the time that those on either side of it are in operation. Instead of the arrangement described, two sluices are sometimes connected with each battery, and, in such cases, one is being cleaned up while the other is in use. The blankets on the lower sluices, D, D′, often remain some hours without being washed. The gold retained in the battery, added to that collected on the blankets, will, in the majority of cases, amount to eight-tenths of the total produce from the rock operated on. A notable proportion, however, escapes over the blankets, and other contrivances, such as amalgamated plates and concentrators, are adopted to arrest the largest possible percentage of this light gold. For this purpose amalgamated copper plates are generally employed.

*Amalgamated Plates.*—In fig. 223, the discharged material from the blanket-sluices, which are inclined at an angle of about three and a half degrees, is conducted through troughs, E F, E′ F′, lined on the bottom with amalgamated copper plates. From these the current passes to the reservoirs for the collection of tailings. The riffle-plates in the sections, E, E′, are made to slide in and out of their places, for the purpose of being cleaned or re-amalgamated, and are usually from eight to ten inches in length; those in F, F′, are also movable, but are made somewhat longer.

In many American mills erected for the treatment of material containing free gold, that metal is now partly collected on amalgamated plates set within the mortar-box, and partly on plates laid in front of it.

These copper plates are often plated on one side with silver. The front plates are ⅛ inch in thickness, 4 feet 6 inches wide, and vary in length from 6 to 12 feet. The inclination given to the amalgamated plates is regulated by the class of pulp to be treated. In ore composed of quartz, finely divided gold, and sulphide of silver, a fall of 1½ or 1¾ inch per foot is held to be sufficient.

The presence in the water of grease, derived from the machinery or any other source, materially interferes with the process of amalgamation; but the occasional introduction into the battery of a shovelful of wood-ashes from the ash-pit of the boiler, removes any traces of oil, and keeps the surfaces and amalgamated plates constantly bright.

The addition of a minute quantity of metallic sodium, or of a little sodium amalgam, to the mercury employed for the collection of gold has been recommended, and at least two patents have been taken out for the use of sodium amalgam. Dr. Wurtz applied for an American patent in November 1864, and Mr. Crookes made a similar application in this country in February 1865.

The evidence obtained with regard to its efficiency has been of a somewhat conflicting nature, and its adoption by those employed in the treatment of gold quartz has perhaps been less general than was anticipated. It is, however, certain that the addition of a small amount of sodium amalgam to the quicksilver used for separating gold from auriferous pyrites, is sometimes attended with beneficial results.

*Cleaning-up.*—The stamping-mill is kept continually at work day and night, and the frequency with which the battery-boxes are cleaned up is to a great extent regulated by the richness of the rock operated upon. When mercury is introduced into the battery the boxes are cleaned up every three or four days, but, in establishments where no quicksilver is used in the mill, this operation is generally put off until the end of the week. In the former case a very large proportion of the gold is taken from the battery in the form of amalgam, and, even when quicksilver is not introduced, the cleaning up of the battery-box furnishes a considerable percentage of the produce. When it has been determined to clean up a battery, the props are placed under the tappets, so as to keep the stamp-heads raised to their full height; the screens are now removed, and the dies taken out. The whole of the sand and other auriferous material is then carefully collected in a pan, and, after everything that may be adhering to the dies has been washed off into the same vessel, they are again introduced into their places. When quicksilver has been employed in the battery, in addition to removing the dies and collecting the sand and amalgam, the mercury in the riffles is changed and the copper plates scraped and re-amalgamated. When this has been done the screens are fastened in their places, and the props taken from under the tappets. The coarser the gold in the rock treated, the larger will be the percentage of the total produce retained in the battery. In order to separate the gold and amalgam from the sand and pyrites with which they are mixed, the materials resulting from a cleaning-up are generally washed by panning. This may be done in the cistern used for washing blankets, since by this means any gold that may pass over the edge of the pan is collected for subsequent treatment. In a large number of the more modern mills, broad amalgamated copper plates, wells, and riffles, precede the blankets, and the latter are often replaced by more perfect mechanical methods of concentration.

*Amalgamation of Blanket Washings, &c.*—The recovery of the gold contained in the pyrites and other matters caught on the blankets, and collected in the washing tank, is effected by amalgamation in various ways. In some cases they are ground with water and mercury in an arrastra, the bottom of which is an iron pan, to which motion is imparted from shafting connected with the stamping-mill. Sometimes one of the pans described, when treating of the processes employed in Nevada for the amalgamation of silver ore, is used; but in the Grass Valley district, a very simple contrivance is sometimes employed.

This consists of two horizontal wooden rollers, each eight inches in diameter and two feet in length, furnished with numerous flat blades of iron arranged radially on their circumference.

These rollers are, by means of belts, made to revolve in shallow cisterns of mercury, in a direction contrary to that of the current of water flowing through the machine, and above them is a hopper, into which is introduced the sand to be washed. Below the rollers is a riffle-board, with an inclination of seven degrees, either covered with removable plates of amalgamated copper, or, if the latter are not used, having the steps of the riffles reversed and filled with mercury. The auriferous sand taken from the cisterns in which the blankets have been washed, is placed in the hopper, and a stream of water slightly warmed is allowed to fall into it in such a way as to gradually wash it under the rollers, and thence over the surface of the riffle-board.

In some mills the whole of the pyrites, &c., collected on the blankets is ground in iron pans, while, in exceptional cases, where the rock is rich and contains "base metals," every portion of the material issuing from the stamping-mill is subsequently treated in large pans in the same manner as silver ores.

*Tailings, &c.*—The tailings which escape over the blankets and other contrivances employed for arresting gold, are collected in settling pits, and are washed for the purpose of concentrating the auriferous sulphides. This may be effected by the use of tyes, buddles, rockers, shaking-tables, &c., and a valuable amount of gold, that would otherwise be lost, is thus recovered. In many instances the use of settling pits and reservoirs is dispensed with, the concentrating appliances being so arranged that the tailings pass directly on to them from the sluices and riffles.

CONCENTRATORS. — The gold distributed in vein-stone frequently presents peculiar difficulties to the mill man. No matter how perfect may be the appliances a greater or less loss will always occur. In some instances less than 50 per cent. of the assay value of the gold is obtained, while in the most successful cases it scarcely exceeds 90 per cent. Coarse gold enclosed in any kind of stony matrix is collected without much difficulty, but heavy losses are always experienced in the treatment of quartz, in which the gold exists in association with sulphides and arsenides, when concentrating machinery must be employed for collecting and delivering these minerals in a state suitable for their subsequent treatment for gold.

In order to obtain the best results, the grains of which the pulp is composed should be first classified into particles of approximately equal dimensions. The apparatus necessary for this purpose is usually of simple construction. Rittinger contrived the *Spitzkasten* or pointed box, as well as a slime separator, known as the "Spitzlutte," or "V trough," while various German and Belgian engineers have constructed classifiers of various descriptions. In Australia, end-blow shaking-tables, and concave buddles fitted with a special appliance for washing the concentrates while on the floor, are much used; while in America, revolving belts, known under different names, and concentrators of the buddle type, are likewise employed. The Frue vanner is in America extensively employed for concentrating auriferous iron and copper pyrites, as well as for mixed ores carrying small amounts of gold and silver. The revolving belt of this machine, as well as the method of distributing and dividing the pulp, were introduced by Brunton, into Cornwall, for the treatment of tin slimes, more than fifty years ago, the short swinging transverse movement given to the belt being the only new feature of the apparatus. The belt, made of india-rubber, supported on light rollers, moves against the flow of pulp, which is divided by jets of water into concentrates and tailings.

RETORTING, AND FUSION INTO INGOTS.—The amalgam collected during the various operations for the treatment of auriferous products is, similarly to that obtained from placer sluices, first filtered, either through canvas or buckskin, and afterwards retorted, the gold being finally melted into bars.

Generally, the redundant mercury is separated by filtration through a prepared skin, in which the pasty amalgam remaining is wrung, until it assumes the form of a somewhat granular mass, having the consistency of putty. This contains about 35 per cent. of gold, and before being introduced into the retort is moulded into lumps. The retort employed is of cast-iron, and has often the form of an ordinary black-lead crucible, varying in size in accordance with the quantity of amalgam treated at each operation. The top is turned flat, and provided with a well-fitting cover, secured in its place by a screw-clamp. Into this is screwed an inch gas-pipe with a bend at right angles, and, at a distance of about three feet, this is again bent downwards, so as to form another right angle.

Before introducing the amalgam, the inside of the retort is slightly covered by a thin coating, either of clay made into a thin paste with water, or with a mixture of water and wood-ashes. This is done in order to prevent the adhesion of the gold, in case of its being accidentally too strongly heated. The cover should be luted with a little clay, before being fastened in its place. When the balls of amalgam have been put in, and the cover has been fixed in its place, the retort, with its contents, is introduced into an ordinary wind furnace, like that employed for the assay of iron ores, and a coke or charcoal fire made around it. The open end of the pipe will now be within a short distance of the floor, and beneath it is placed a pan of water, into which a piece of canvas, bound around it so as to form a hose, is allowed to dip to the depth of about half an inch. In order to prevent accidents from the ascent of water into the retort, the level of that in the pan must be kept below the end of the metal pipe, and the descending limb of the apparatus is cooled by being bound with cloths which are wetted by the constant application of cold water, or a Liebig condenser is sometimes employed. The latter has the advantage of being neater, and also of requiring less attention on the part of the person in charge of the operation. When the apparatus has been thus arranged the fire is lighted, and the heat gradually increased, until the retort has acquired a dull-red colour, care being at the same time taken to insure perfect condensation of the mercury. In this way the heat is kept up for some time, but when the pipe begins to cool, and drops of quicksilver no longer fall from its extremity, the operation is finished. The fire may now be withdrawn, and the retort removed from the furnace. The cover should not, however, be removed until the retort has become nearly cold, to avoid the diffusion of mercurial vapours in the atmosphere. When very large quantities of auriferous amalgam have to be dealt with, a fixed retort, similar to that employed for the treatment of silver amalgam, may be used with advantage.

Retorted gold is generally melted, for the purpose of being cast into ingots, in the furnace used for heating the retort during the distillation of amalgam. Either coke or charcoal may be employed as fuel, and the black-lead pots in which the fusion is effected should be well annealed before being exposed to the full heat of the fire. The spongy gold, which at the commencement of the operation filled the pot, when fused, occupies much less space than it did previously; so that as soon as the first charge has melted, the cover may be taken off, and a further addition of retorted gold made. A little borax is added with each charge, and when the crucible has become sufficiently full of fused metal, it is withdrawn with a pair of stout tongs, and its contents poured into open cast-iron moulds.

The cost of extracting the gold from a ton of auriferous quartz varies within wide limits according to the richness and nature of the rock, and the cost of the fuel, labour, &c., in the locality. It is also evident that water-mills can work more economically than those employing steam-power. In California, in 1875, the Chariot Mine, San Diego, treated

2,500 tons of quartz, yielding on an average $55, at a cost of $4 per ton of 2,000 lbs., while the Sierra Buttes, in Sierra County, worked 53,959 tons, yielding on an average $8·50, at a cost of $0·83 per ton. During the same year the average cost of treating one ton of auriferous quartz at thirteen of the most productive mines in California was $1·86 per ton.

At the present time (1886) the cost of milling is considerably lower than in 1875. At Sierra Buttes, with ninety-three heads of stamps at work, it is reduced to $0·56 per ton.

At the Home-Stake Mines, Dakota, auriferous slate is amalgamated within the mortar-boxes and on plates in front of them. During the year ending 1883, the number of heads in work was 200; number of tons crushed, 191,505; value of stuff, 23s. 9d. per ton; cost of mining and milling, 16s. 5d. per ton; profit, 7s. 4d. per ton.

*Dry Stamping.*—Auriferous ores, if largely associated with sulphide of silver, are usually first dried in a revolving furnace, then stamped dry in the mill with a proportion of common salt, and subsequently chloridized in a revolving or Stetfeldt furnace. A head weighing 800 lbs., with a drop of 7 inches, making ninety drops per minute, will pass through double discharge wire-wove screens having nine hundred holes per square inch, about one ton of dry dust during twenty-four hours. With a single discharge under similar conditions of speed with the use of water, a head will reduce to the condition of pulp of the same degree of fineness about twice as much ore in the same time.

## AUSTRALIAN GOLD MILLS.

In Australia the reduction of auriferous quartz is effected by methods generally similar to those described as in use in California; but there are some notable differences in the construction of the machinery, as will be seen from the following account of the Australian gold-saving apparatus in use at the Phœnix Gold Mines, in South Wynaad, India. The quartz at that mine is of the usual degree of hardness, and is associated with from 2 to 3 per cent. of pyrites assaying from 2 to 4 ounces of gold per ton. The quartz itself contains only from 3 to 5 pennyweights per ton. The quartz is reduced to a pulp which passes through screens with two hundred holes to the square inch. The pulp and water flow over three sets of amalgamated copper plates with a fall of 1 in 14, whose width is the same as the length of the mortar-box, namely, 50 inches. Fig. 224 is an end elevation of the stamps employed.

Contrary to American practice, the entire framework, guides, and guide-pieces are of cast-iron, while the cams are on a second motion shaft driven by spur gearing. A′, cast-iron frame; B, platform in front and rear of cam shaft; C′, spur-wheels for driving cam shaft; D, steel cams mounted on shaft; F, mortar-box; G, stamp head; H, die at bottom of mortar-box; J, stamp screen; K, splash cover of sheet-iron hung in front of stamp screen; L, flat screen for distributing pulp equally across the head of first amalgamated plate; M, first amalgamated plate, 24 inches long, on the line of flow; N, second amalgamated plate, 63 inches long; O, third amalgamated plate, 24 inches long; P, mercury well; R, pipe for supplying mortar-boxes with water; S, weight for keeping clutches in gear; T, stem attached to stamp-head.

It will be observed that the top of the die is level with the bottom of the screen, and that the distance between the stamp-head and front of the mortar-box is very small, about 1¾ inch. The amalgamation of gold with mercury is not effected within the mortar-box, but takes place exclusively on the plates and in the mercury well. From the mercury well,

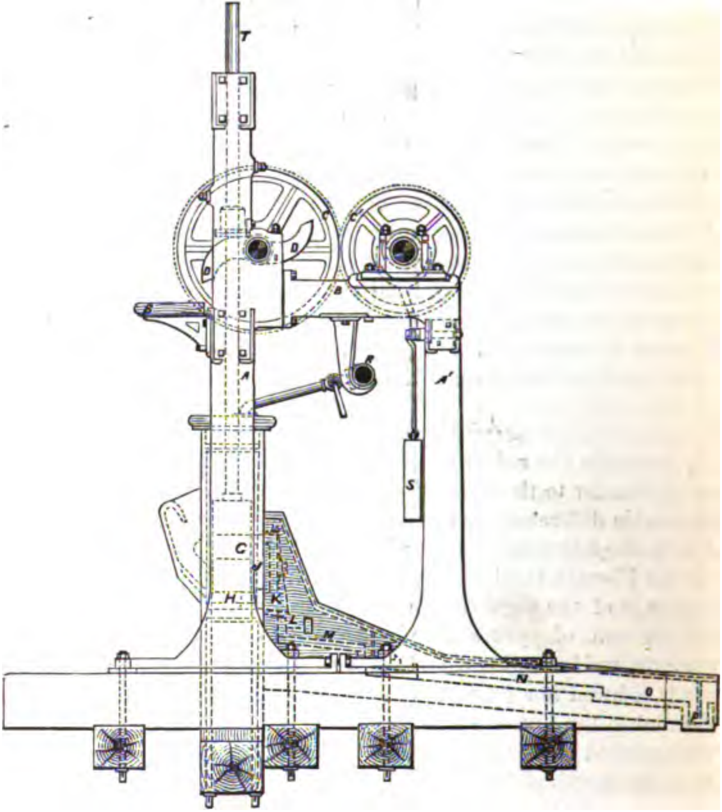
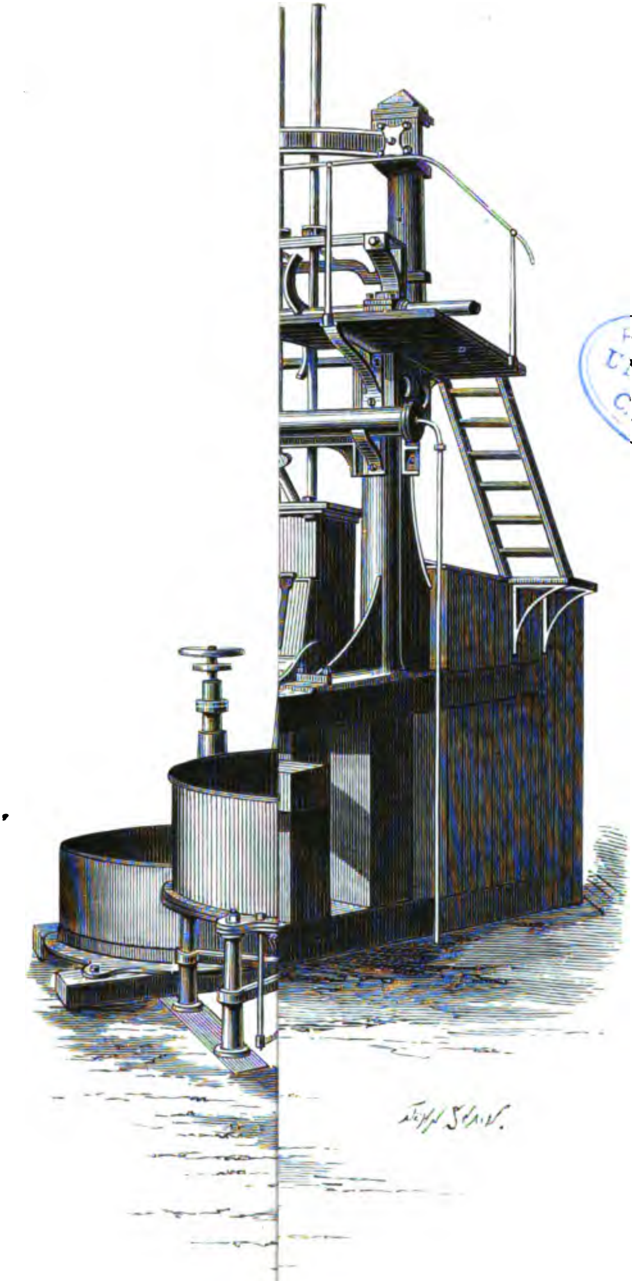


Fig. 224.—Australian Stamping-Mill; end elevation.

P, the pulp passes over amalgamated plates and two mercury riffles, each about 3 inches wide and 1½ inch deep, and is finally concentrated on a shaking-table having at the head a pan-like depression from the bottom of which the concentrated sulphides escape, through an easily regulated valve, and are collected in a reservoir, while the impoverished tailings pass off at the other end. The pyritic concentrates from the shaking-table are subsequently amalgamated in Wheeler pans and finally passed



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through a set of Hungarian mills. Each stamp-head makes from 60 to 70 drops, each of 9 inches, per minute.

From the reduction of 7,500 tons of quartz, 270 tons of pyritic concentrates were obtained. The gold collected from the pyritic quartz, 7,500 tons, was 1,075 ounces, or at the rate of $\frac{2}{10}$ dwt. per ton.

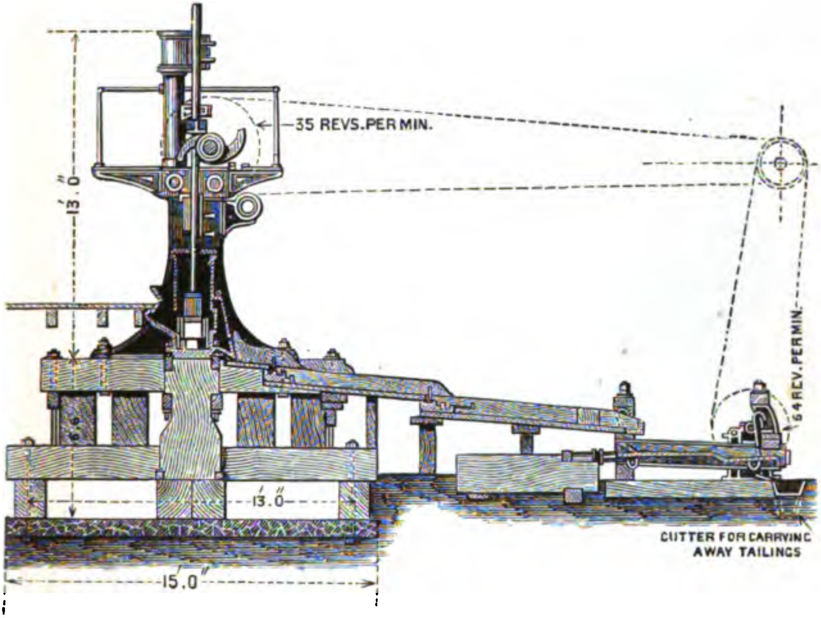

Fig. 225.—Queensland Gold Mill; longitudinal section.

The several parts of the apparatus afforded the following proportionate quantities of the total weight of gold obtained :—

| | Per Cent. |
|---|---|
| Mortar-boxes | 9·2 |
| Amalgamated plates | 55·8 |
| Mercury well and riffles | 8·1 |
| Wheeler pans (from concentrates) | 22·2 |
| Hungarian mills do. | 4·7 |
| | 100·0 |

The amalgam collected on the plates contained about one-third of its weight of gold. The concentrates, 270 tons, yielded 1 oz. of gold per ton.

An exceptionally well-constructed quartz-crushing and amalgamating mill, erected for the Queensland Commissioners at the late Colonial and Indian Exhibition by Messrs. John Walker & Co., of Maryborough, Queensland, is represented in perspective elevation in the folding plate,[1] and in transverse section and plan in figs. 225, 226, which have been kindly furnished by the proprietors of 'The Engineer.' The stamp battery of five heads, each weighing 800 lbs., and capable of crushing 48 cwts. of quartz in twenty-four hours, has a cast-iron framing of which the principal element is a 10-inch tubular column, with a flanged base 7½ feet wide, carrying the bearings of the cam shaft, as well as a lighter pair of pillars with cross girders for the stamp-guides. The cam shaft, driven at one end by a belt from the engine, makes about 35 revolutions, giving double that number of drops of the stamps, per minute. The stamp-dies are placed at most two inches below the lower sill of the discharging aperture. The screens are of best charcoal sheet-iron, about No. 28 B.W.G., with 225 holes to the square inch. Chilled cast-iron shoes and dies are used; but according to Mr. Longsden, very good results have been obtained in Queensland from the use of wrought-iron shoes made from scrap blooms, as the soft metal becomes incrusted with particles of quartz, and then acts much in the same way as a diamond- or emery-faced surface. The battery water is supplied at the rate of 100 gallons per hour for each head from the pipe below the cam shaft in front. Mercury is used in the battery, and a strip of blanket is placed along the sill of the discharging aperture, immediately in front of the screen, to prevent loss of amalgam. The surface amalgamators, 5 feet in width, are divided into three series by two sets of mercury wells and riffles, whose lengths are 22 inches, 6 feet, and 7½ feet respectively. The mercury well is a square trough, provided with an amalgamated stop or back plate, which brings the stream falling over the edge of the board into close contact with the mercury. The copper plates are electro-silvered before amalgamation, and sodium amalgam is used at regular intervals to quicken the action as well as to prevent flouring of the mercury. The concentration of the pyrites is effected by a percussion-table with a continuous discharge placed below the surface amalgamator. This is horizontal at the upper end, but sharply inclined towards the bottom, and is also slightly dished in the centre, having a discharging aperture at the lowest part, which is regulated by an adjustable valve. The table is suspended by links at the four corners, and is kept in rapid movement by a three-armed cam, making 64 revolutions per minute, which forces it against a buffer-stop placed below the lowest

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[1] The battery is at rest, with the stamps "hung up."

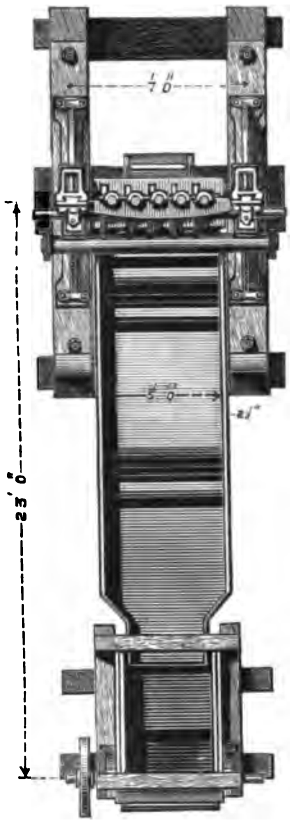

Fig. 226.—Queensland Gold Mill; plan.

amalgamating table. The heavy deposit passes through the valve, while the lighter waste is carried over the end of the table into the tailings gutter below. The concentrated deposit, after passing through a Boss pan, where it is triturated with mercury, is led into a 7-foot settler, where the last separation of amalgam takes place. At a subsequent period, after the illustration was engraved, two Berdan pans were added to the plant. In these machines the grinding is effected by heavy balls of cast-iron, working in an annular trough of U-shaped section, which revolves about an inclined axis. They are used to a considerable extent in Australia, but not much elsewhere.

### Hungarian Mill.

At Schemnitz in Hungary, quartz vein-stuff, containing iron pyrites, galena, and a little free gold, is stamped and subjected to amalgamation in the apparatus represented in fig. 227, known as the Hungarian or Tyrolese mill. This consists essentially of a cast-iron basin, *a*, *b*, *c*, *d*, carried upon a framework, A, with a driving-shaft, *x*, passing through a tube in the centre, which receives motion from a spur-wheel below, and turns the runner, *f*, by means of a cross head and suspension rods, *g*. The outside of the runner, which is made of hard wood, is similar in shape to the inside of the pan, and carries a series of iron blades projecting radially from the less steeply inclined part of the side; and it also has similar blades on the bottom. The inside is hollowed out to a funnel shape, leaving a narrow annular passage for the admission of the pulp around the central tube. The bottom of the basin is filled with mercury to a depth of ½ to ¾ inch, which requires about 50 lbs. in a 16-inch pan. Usually several mills are employed in connection with the same stamp-battery, and are arranged in series at different levels, as shown in fig. 227. When the runner is so adjusted by the suspension rods that the blades are about ¾ inch from the surface of the mercury, it is set in motion, and the pulp from the stamps, arriving by the spout S, enters at the centre of the upper mill, and after passing over the mercury, is discharged by S′ into the lower mill, where it is similarly treated, and passes out by S″ to the concentrating apparatus. The speed of rotation varies from twelve to

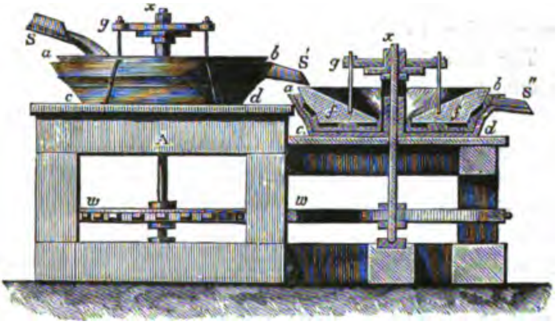

Fig. 227.

twenty-four revolutions, according to the density of the associated minerals, and is so regulated that the current may pass freely over the mercury without forming any deposit upon it, the flow being kept up by the centrifugal action of the runner blades, so that only the free gold subsides and amalgamates, while pyrites, galena, and other minerals pass out together with the gold that escapes the action of the mercury. The mills are cleaned out at intervals of two to four weeks, and the amalgam is filtered and distilled in the usual way.

According to Rittinger, a larger proportion of gold is saved when distilled mercury is used in the mills instead of that squeezed out of the amalgam in filtration, as the latter, although containing only from 4 to 6 grains of gold per lb., is less efficacious in amalgamating than the pure metal; but the increased cost and trouble due to the necessity of more frequently changing and distilling the mercury prevents this method of working from being practically adopted. The vein-stuff treated yields at the rate of about 1 lb. of gold per 100 tons, or about 3½ dwts. per ton, in addition to what may be recovered by smelting the concentrates. The loss of mercury is from 2 to 4 lbs. per 100 tons, but may exceptionally be as high as 10 lbs. if the slimes contain clay or antimonial ores. When three series of mills are used, 65 to 75 per cent. of the total gold is saved by the upper, 20 to 25 per cent. by the middle, and 6 to 10 per cent. by the lower series. A mill will treat about 18 cwts. of stuff in twenty-four hours, and requires about $\frac{1}{25}$ of one-horse power to drive it.

This apparatus, though comparatively restricted in its application, is used to some extent in Australian gold mills (p. 785). Various attempts have been made to extend the same principle, namely, that of centrifugal distribution, to pumping or forcing slimes through a deep column of mercury, in order to obtain more perfect amalgamation; but most of these contrivances have the disadvantage of producing a great waste of mercury.

## Amalgamation of Pyrites at St. John D'el Rey.

The rock enclosing the Morro Velho lode is a clay-slate of tolerably uniform texture, the lode itself being strong and well defined; but irregular in direction, dip, and dimensions, and frequently traversed by fissured clay-slate, or barren white quartz. The valuable part of the vein-stuff, composed of quartz with iron pyrites disseminated throughout, is subjected to mechanical concentration in order to collect the pyrites for barrel amalgamation, but the use of mercury either in the battery or on surface-plates is dispensed with. The order of the different operations is as follows:—

Stamping.—The ores on being drawn from underground are reduced by stone-breakers to a proper size for the stamping-mills, which are of the ordinary Cornish and Californian patterns, and are worked by water power. The screens used are pierced with conical holes $\frac{1}{12}$ inch in diameter on the outside, tapering off on the other to $\frac{1}{48}$ inch, where a projecting burr is raised towards the inside of the battery-box. Some experiments made for the purpose of ascertaining the yield of gold incident to deep and shallow stamping afforded the following results:—

| | Depth of Die below Bottom of Screen. | Loss of Gold per Cent. | Sand Produced not passing through 120 Holes to Linear Inch. | Tons Stamped per Day. | Number of Drops of Head per Minute. |
|---|---|---|---|---|---|
| | Inches. | | Per Cent. | | |
| Deep stamping . . | 20 | 30·96 | 12½ | 1·3 | 60 |
| Shallow . . . | 6 | 44·70 | 28 | 1·8 | 60 |

The loss of gold in shallow stamping was 44·7 per cent., while in deep stamping it was reduced to 30·96 per cent., or about 14 units in favour of the latter.

The number of stamp heads at work in May 1885 was 132. The work performed by these heads for twelve months to that date is given as 1·27 tons per head daily.

Concentration by Strakes.—The discharge from the screens in front of the battery-boxes is diluted with a certain amount of clear water, and conducted over inclined strakes each 20 inches in width and 22 feet in length, which have a fall of 1 in 10. These strakes are made of well-seasoned 1½-inch planks nailed to triangular frames. The three surfaces or strakes are carefully covered with coarse sail-cloth tarred on the under side and then fastened with tacks, while on the edges of the sides are screwed slips of wood. Each end of the strake-frame is supported on a wrought-iron axis, and as soon as it is necessary to present a fresh concentrating surface, the upper strake is turned one-third of a revolution, when the material already concentrated and resting on its face is washed off by means of a water-jet into a V-shaped trough beneath. This pulp then flows to a catch-pit. A pair of strakes is represented in cross section on A B, and end elevation, in fig. 228, in side elevation in fig. 229, and in plan in fig. 230. *a*, launders, bringing the pulp and water from stamps to head of strakes; *b*, bottom of strake covered with sail cloth; *c*, side slips clipping the edges of the sail-cloth and keeping the pulp on the strake table; *d*, triangular frame on which the strakes are laid; *e*, catchbox; *f*, waste launder for tailings; *g*, concentrate launder leading to the catch-pit; *h*, wrought-iron axles on which the strakes are suspended; *k*, laps of canvas to make up joints. For a battery of thirty heads of Californian stamps reducing about 40 tons of quartz per day of twenty-four hours, twenty-four such strakes are necessary.

The question of increasing the weight of auriferous concentrates for barrel amalgamation, and reducing the loss of free gold in the treatment of the pulp from the screens, has received serious and careful attention. In an experiment with a new skin and woolly baize conducted under similar conditions, it was found that the proportion of sand retained on the skin was to that on the baize as 1 to 5·5, while the proportion of gold collected was in the inverse ratio, namely, as 1 to 0·52, a result unfavourable to the use of baize. In a second series of experiments a hairless skin was first placed on the strake, then woolly baize, followed by common baize. The proportional figures for sand and gold caught were :—

| | Hairless Skin. | Woolly Baize. | Common Baize. |
|---|---|---|---|
| Sand . . . . | 1 | 51·0 | 12·0 |
| Gold . . . . | 1 | 0·3 | 0·31 |

The inferiority of woolly baize to other materials for collecting gold was thus rendered so apparent that it was withdrawn from use at the straking floors.

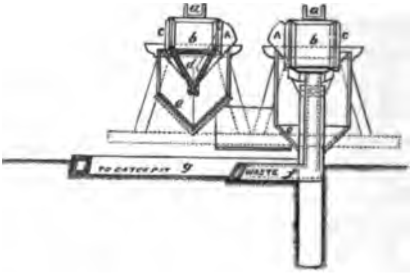

Fig. 228.—Pair of Strakes ; cross section on A B, and end elevation.

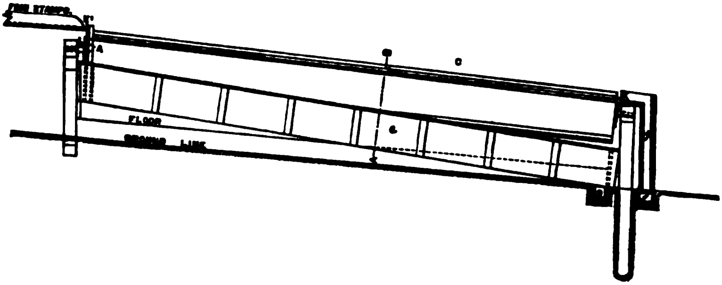

Fig. 229.—Pair of Strakes ; side elevation.

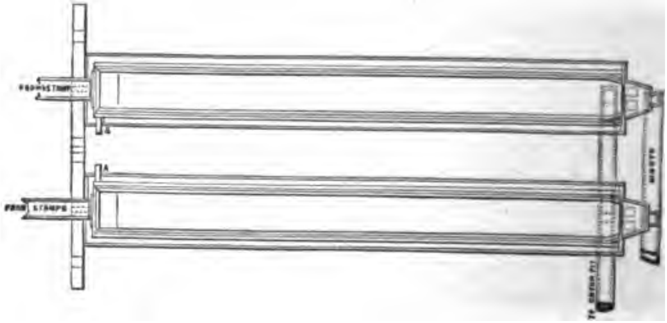

Fig. 230.—Pair of Strakes ; plan.

Amalgamation of Concentrates.—The concentrates obtained from the revolving strakes are generally amalgamated in barrels and not in pans, the use of the latter being of somewhat recent date. Each barrel takes 16 cubic feet or 14 cwts. of concentrates, measuring 22 cubic feet to one ton, and 50 lbs.[1] of mercury. The barrel, charged with concentrates and water only, is rotated for a period of fifteen minutes, when it is stopped and opened to test whether the contents are in proper proportions. If too much water is present the mercury sinks to the bottom, and will not mix with the concentrates; but if too little is employed the concentrates cut and flour the mercury. The consistency of the pulp should be that of thick honey, which is readily ascertained by dipping a stick into it. If the stuff slides off too freely it is either too dry or too wet, and either water or concentrates must be added as required. When this is corrected a few more turns of the barrel are made, and the mixture is again tested. When the proportions of sand and water are found to be satisfactory the mercury is introduced, and the barrel is turned for twenty-four hours at a speed of fourteen revolutions per minute. The contents of the barrel are then discharged into an apparatus known as a "saxe," which has for its object the removal of slime and the collection of the amalgam and globules of quicksilver. This consists of a series of rectangular boxes (*caixas*) containing from 2 to 3 cwts. of mercury at the bottom, with a number of agitating blades which receive a reciprocating motion from the same wheel that turns the barrels. With the view of further diminishing the loss of amalgam and mercury, the pulp from the saxe is passed over a system of strakes, and the concentrates, after grinding in arrastras, are again amalgamated. The loss of mercury is commonly reckoned at about ⅛ ounce per cubic foot of concentrates; but the actual loss experienced on 3,595 tons amalgamated amounted to 12¾ ounces per ton.

The following statistics relate to working results during ten years ending May 1885:—

| Year. | Total Number of Tons of Mineral raised from Shafts. | Total Number of Tons of Mineral Stamped. | Total Tons of Sand Concentrated and Amalgamated. | Loss of Mercury on Sand Amalgamated, Lbs. | Average yield of Mineral after Treatment by the Stamps, in oz. Troy per Ton. | Loss of Gold on Treatment. | Cost per Ton of | |
|---|---|---|---|---|---|---|---|---|
| | | | | | | | Mineral Stamped. | Mineral Received from Mine. |
| | | | | | | Per Cent. | s. d. | s. d. |
| 1876 | 63,693 | 62,474 | 2,944 | 2,377 | 1·110 | ... | 3 9 | 3 8¼ |
| 1877 | 66,127 | 63,058 | 3,343 | 2,184 | ·812 | 25·00 | 3 10 | 3 7¾ |
| 1878 | 67,915 | 66,976 | 3,631 | 2,420 | ·834 | 29·20 | 4 0¼ | 4 0¼ |
| 1879 | 69,954 | 69,548 | 4,073 | 1,822 | ·690 | 31·50 | 3 6¼ | 3 6¼ |
| 1880 | 62,681 | 63,540 | 3,887 | 1,817 | ·727 | 26·50 | 3 11¼ | 4 0¾ |
| 1881 | 46,164 | 45,564 | 3,097 | 1,907 | ·640 | 24·36 | 5 0⅛ | 4 11¾ |
| 1882 | 67,463 | 63,939 | 3,372 | 1,671 | ·555 | 31·35 | 3 9½ | 3 7¼ |
| 1883 | 72,496 | 70,142 | 3,863 | 2,605 | ·381 | 29·14 | 3 8½ | 3 6⅛ |
| 1884 | 69,254 | 64,940 | 3,771 | 3,796 | ·358 | 34·42 | 3 8½ | 3 5¾ |
| 1885 | 61,230 | 60,065 | 3,970 | 7,815[2] | ·376 | 33·21 | 3 8½ | 3 7¾ |
| | 646,977 | 630,246 | 35,951 | 28,414[3] | | | | |

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[1] See 'Mining and Metallurgy of Gold and Silver,' p. 218.

[2] Excessive loss through leakage of apparatus.

[3] Per ton of sand amalgamated = 0·79 lbs.

On referring to the foregoing table, it will be observed that every hundred tons of quartz reduced to pulp, gave $5\frac{7}{10}$ tons of concentrates, which were amalgamated with an average loss of $12\frac{5}{10}$ ounces of mercury per ton; also that the gold obtained varied from $65\frac{5}{10}$ to $75\frac{6}{10}$ of the total quantity contained in the mineral.

The net profit on milling and treating 630,246 tons of ore was £524,727, or 16s. 7½d. per ton.

The Morro Velhó Mine was purchased in 1835, and during a period of fifty years to June 1885, yielded gold to the amount of £5,059,501.

### ARRASTRA AMALGAMATION AT PESTARENA.

The Pestarena Mines include those of Val Toppa, situated in the Val Ossola, and of Pestarena, about eighteen miles distant from the former, in the Val Anzasca, in North Italy. The general characters of the deposits are given in the following table:—

| | Val Toppa. | Pestarena. |
|---|---|---|
| Rock enclosing reef. | Micaceous and talcose slates occasionally occurring together. | Micaceous schist. |
| Gangue. | Almost entirely quartz. | Micaceous schist associated in varying proportions with quartz. |
| Ore. | Auriferous iron pyrites with mispickel, containing nickel in very small quantities. | Similar to that found in the Val Toppa Mine. |
| Native gold. | Rarely found. | Never found. |
| Character of gold. | Gold very minutely divided: gravitating property very feeble. | Gold similar to Val Toppa. |
| Special difficulty in mechanical treatment. | Gold more or less enfilmed in micaceous schist, which protects it from amalgamation. | Same as Val Toppa. |

Considerable attention has been given at these mines to the determination of the value of various reducing and amalgamating apparatus. Stamps and amalgamated plates have been tried with the result that only 65 per cent. of the contents of gold in the stuff was obtained; but with the modified form of arrastra known as the Francfort mill, about 82 per cent. of the gold is secured. It appears that time and attrition for brightening the particles of gold, are requisite for realizing the higher result.

The ore and gangue, when brought to surface, are first washed on tables; the large pieces are then reduced by a stone-breaker, and the stuff from both operations is dry crushed by rolls, and sized by a drum-sieve with twelve holes to the square inch into particles of 6 mm. and less, which are sent to the Francfort mills.

FRANCFORT MILL.—This mill is represented in section in fig. 231, and in plan in fig. 232. The outer shell of wood, *a*, 10 feet 8 inches diameter at top, and 10 feet 3 inches at bottom, is strongly hooped with iron; *b*, mill runners, four in number, 30 inches wide, 10 inches thick, and about 25 inches long; *c*, ring of wrought-iron, 4 inches wide and 1¼ inches thick, attached to runners by wrought-iron staples, *d*; *g*, screws, 2½ inches diameter, placed midway between runners for lifting or lowering the wrought-iron ring, *c*, and runners, *b*; *x*, inner shell of wood, 5 feet diameter at bottom, and 4 feet 9 inches at top, kept together by wrought-iron hoops on the outside; *y*, spinning launders; *t*, hole in bottom of mill bed, plugged with hemp when in work; *w*, launder for receiving pulp and amalgam at the time of the "clean up," when the hemp plug is withdrawn; *v*, spur wheel by which the mill is driven; *t*, hand-wheel for throwing the friction cone, *s*, in or out of gear. By means of the chains and shackles shown in fig. 232, the "drag" of the runners or mullers is thrown upon the wooden arms attached to a segment piece centred around a vertical shaft 4¼ inches diameter.

The height of the outer shell from the ground is 40 inches, the depth from the upper edge to the grinding and amalgamating floor is 20 inches, and the width of the latter is 30 inches. In charging the mill about 6 inches of water is allowed to flow on the bed, the arms are then set in motion at twelve revolutions per minute, and the first charge of 1½ quintal of stuff is drawn from the storage hopper. The chains connecting the arms with the runner ring are then slackened, and the set-screws on the arms lowered until the full weight of each runner bears upon the stuff. In this position the runners remain during the grinding and amalgamating operation. When it is seen that the speed of the mill is gaining on the regulation number of twelve revolutions per minute, the second charge of 1½ quintal of stuff is added. At the same time clean water is supplied until the depth measures from 8 to 10 inches. After the lapse of forty minutes the mill is ready for a third charge; no fresh water is, however, added on this occasion. This third charge, completing the first series of charges, gives a total of about five quintals dry weight, while the time occupied in the first stage of grinding the stuff is two hours. The speed of the stones is now reduced to eight revolutions per minute. At the close of another hour mercury is added to the pulp, the quantity being three kilogrammes per mill. The time of the delivery of the mercury as well as the weight employed is, however, regulated by the class of ore under treatment, pyrites containing arsenic or other deleterious substances necessitating a prolongation of the operation until the stuff is reduced to a very fine paste. In a case of this kind the mercury is not delivered until an hour later than is usual with ordinary ore. If the mercury "flours" and floats on the surface of the water, four kilogrammes of dry fresh lime are thrown into the mill. One hour and a half after the ordinary time of adding the mercury to the charge, the speed is lessened

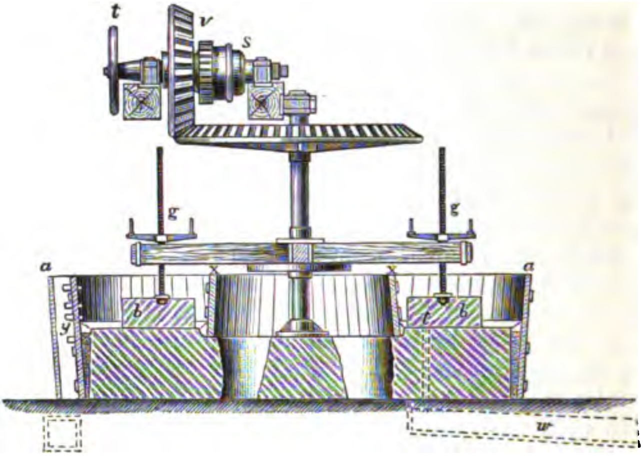

Fig. 231.—Francfort Mill; vertical section.

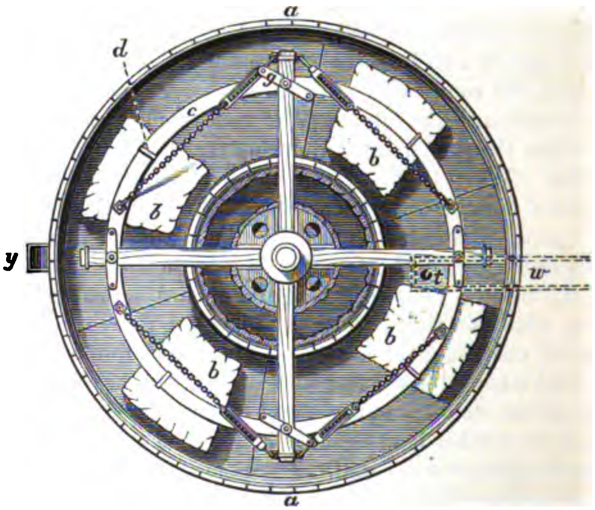

Fig. 232.—Francfort Mill; plan.

to six revolutions, and two hours later it is reduced to five revolutions per minute. At this point in the operation 6 inches of water is added, making from 16 to 18 inches of standing water in the mill. This rate of speed, five revolutions per minute, the lowest used, is maintained for two hours. The process termed "spinning" now takes place, that is, about 6 inches of the top water is run off from the mill, leaving 12 inches still on the bed. To effect this the plugs in the outer shell within a vertical launder are removed, the highest first, in order to avoid any mixture of the heavier with the finer material.

The mill has now been working twelve hours, or one half the time that constitutes a "clean up," and the water having been "spun off," it is now ready for the second series of charges, which is divided into three periods as before described. The next twelve hours are a repetition in details of the preceding twelve; the only difference being that the charges are not so heavy, while the first series represents 5 quintals, the second series consists of but 2½ quintals; the total in twenty-four hours being 7½ quintals, dry weight, of stuff treated per mill.

The following abstract statement distinguishes the various operations connected with the working of a charge, and gives the time when the speed of the mill is changed, also when delivery of mercury to the charge is made. It is assumed that the mill has been cleaned up at two, and restarted with a fresh charge of stuff at three o'clock in the afternoon:—

### *First Operation.*

| Time. | | | | | | | Total Time. |
|---|---|---|---|---|---|---|---|
| 3.0 p.m.— | 1st | charge— | Speed, | 12 | revols. per min.— | 6 inches of water in mill | 5 hours. |
| | 2d | do. | do. | 12 | do. | 8 to 10 inches do. | |
| | 3d | do. | do. | 12 | do. | do. do. | |
| 7.0 p.m. | do. | do. | do. | 8 | do. | do. do. | |
| 8.0 p.m. | do. | do. | do. | 8 | do. | Mercury added to charge | |
| 9.0 p.m. | do. | do. | do. | 6 | do: | | |
| 11.30 p.m. | do. | do. | do. | 5 | do. | Water increased to 18 ins. | 6 hours. |
| 1.30 a.m. | do. | do. | do. | 5 | do. | "Spinning," or 6 inches of top water drawn off | |

### *Second Operation.*

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| 2.0 a.m.— | 1st | charge— | Speed, | 12 | revols. per min.— | 12 inches of water in mill | 11 hours. |
| | 2d | do. | do. | 12 | do. | Water slightly increased if necessary | |
| | 3d | do. | do. | 12 | do.. | do. | |
| 6.0 a.m. | do. | do. | do. | 8 | do. | 12 or more inches of water in mill | |
| 8.0 a.m. | do. | do. | do. | 6 | do. | do. | |
| 10.30 a.m. | do. | do. | do. | 5 | do. | Water increased to 18 ins. | |
| 1.0 p.m. | do. | do. | do. | 5 | do. | "Spinning," or 6 inches of top water drawn off | |
| 2.0 p.m. | | do. | Mill cleaned up and amalgam collected—total time, 24 hours. | | | | |

It will be observed that although the operation includes two series of charges, mercury is only added once, at the end of the fifth hour. Clean water is brought into the mill by a supply-pipe running the length of the house. This pipe is provided with cocks, one for each mill. To effect the clean-up of a mill, the hemp plug is removed from the hole in the bed and the contents are run into a launder beneath it and thence to the amalgam trough, from which they are subsequently removed in bowls. In the upper floor of the mill-house a tramway is laid, over which the stuff is brought direct from the crusher and tipped into the store-hoppers. Of these hoppers there is one to each mill having a capacity of two tons weight. A short launder carries the ore from the hopper into the bed of the mill, the weight of charge being regulated by a small door and lever. Large troughs are employed for receiving the tailings, and, when full, are cleaned out. The slime is re-treated in the mill, a little clean quartz being sometimes added to assist in reducing the finer grains. The tailings only remain in the mill twelve hours, when they are removed, and the mill is re-charged.

The mill-beds are of gneiss, this being the best rock procurable in the district for that purpose. The gneiss is, however, at times almost granitic. The bed consists of from six to eight stones. The upper surfaces and end-joints are worked off level with the chisel, the remainder of the stone being roughly trimmed.

In forming a mill-bed great care is taken that the upper surface is perfectly level. The space between the joints is filled with hydraulic cement, while the space between the outer tubbing and the circumference of the bed is filled with sand to within 3 or 4 inches of the top, when hydraulic cement is added to complete the surface. An ordinary mill-bed lasts about ten months. The mill-runners or mullers are made from the hardest close-grained gneissic rock obtainable, since it is found that the crushing is more satisfactorily performed when hard runners are used. The front part of each runner is under-cut, so as to catch the stuff for grinding. A set of four runners last from six to eight weeks.

The auriferous stone is very compact, a greater or less quantity of hard quartz always occurring with the pyrites, and the free gold is very finely divided. The reason, probably, that the Francfort mill gives such high results, is that the stuff being confined and in constant movement under pressure continually presents fresh surfaces to the runners, and the gold comes more continually into direct contact with the mercury than is possible either in battery or in plate amalgamation.

The following statistics are from actual results obtained at the Pestarena and Val Toppa Mines :—

| Mines. | No. of Francfort Mills. | No. of Metric Tons of Stuff Reduced. | Produce of Fine Gold determined by Assay per Statute Ton. | | | Extracted by the Francfort Mill per Statute Ton. | | | Proportion of Total Gold obtained by the Mill. | Loss of Mercury per Metric Ton. |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | Oz. | Dwts | Grs. | Oz. | Dwts | Grs. | Per cent. | Grammes. |
| Pestarena . | 28 | 450–500 | ... | 12 | 7¾ | ... | 10 | 5½ | 82½ | 1,890·6 |
| Val Toppa . | ... | 500 | ... | 6 | 10½ | ... | 5 | 6½ | $81\frac{9}{10}$ | 174·5 |

### Combined Pan Amalgamation and Concentration.

When an ore contains a notable amount of both gold and silver, the Washoe process of amalgamation, combined with some form of mechanical concentration, may often be advantageously employed. This method is adopted at one of the principal mines in America where the vein-quartz includes free gold, in an exceedingly fine state of division, combined with blue and green carbonates of copper, and sulphides of silver, lead, antimony, and copper. The assay value of the ore has averaged gold 80s. and silver 55s. per ton, while the pulp from the settlers after pan amalgamation contained gold 14s. and silver 22s. per ton. The gold obtained in the mill averaged 82·5 per cent. and the silver 60 per cent. of the assay value. The mill contains two stone-breakers, fifty heads of stamps, each head weighing 850 lbs., twenty-four Frue vanners, twenty-four pans, twelve settlers, and five tailings pans. The drop of each stamp head is about 7 inches, the number of drops per minute, 90. The screens, 4 feet long and 15 inches deep, are perforated with 900 holes per square inch. The weight of ore reduced per twenty-four hours is about 105 tons or 2 tons per head. The pans are 5½ feet in diameter, and 2 feet 9 inches deep; the speed of the grinding mullers of these pans is 70 revolutions per minute. The settlers, 8½ feet in diameter and 3 feet deep, are fitted with agitators which make 16 revolutions per minute.

The treatment of the ore is as follows:—It is brought from the mine and passed over grizzlies or inclined screens, which separate fragments varying from impalpable powder up to 1½-inch cubes from the larger pieces, which pass to the stone-breaker, and when reduced in size, together with the product from the grizzlies, drop into hoppers, whence they are delivered to the mortar-boxes by automatic feeders. In the mortar-box the fragments of ore are reduced so as to pass through screen-holes about $\frac{1}{16}$ inch in diameter. A sample of the pulp passed through a series of fine meshed screens gave by weight the following proportionate quantities:—

| No. of Holes per Square Inch. | Approximate Area of Mesh. | Retained in Sieve. |
|---|---|---|
| 900 | ·022 sq. inch. | ·70 per cent. |
| 1,500 | ·016 ,, | 4·66 ,, |
| 2,200 | ·013 ,, | 12·12 ,, |
| 3,000 | ·011 ,, | 5·39 ,, |
| 5,000 | ·0085 ,, | 13·34 ,, |
| 10,000 | ·0085 ,, | 18·61 ,, |
| 14,400 | ·0045 ,, | 9·09 ,, |
| 22,500 | ·0035 ,, | 6·36 ,, |
| Stuff passed through | | 29·72 ,, |
| | | 99·99 ,, |

The pulp from the screens, without classification into grains of specific dimensions, is passed over Frue vanners, of which there are two to each set of five heads of stamps, by which concentrates varying in value from £100 to £200 per ton of 2,000 lbs. are obtained for smelting, as well as tailings for the amalgamating pans. These pans are charged, worked, and discharged every six hours. A charge consists of about 3,500 lbs. of pulp, 60 to 70 lbs. of quicksilver, and small quantities of sulphate of copper and sulphuric acid. The charge is heated direct by live steam, with the view of quickening the process and aiding in the extraction of bullion. From the pans the charge is passed to the settlers, where a separation of the pulp from the amalgam is effected, and from the settlers the pulp is a second time concentrated in twelve other Frue vanners. The amalgam is sent to the clean-up pan and subsequently strained in order to get rid of the free mercury, and to obtain a hard amalgam for the retort. The concentrates are sold to smelters. The weight and cost of chemicals consumed per ton of ore, as well as the cost of milling, may be given approximately as follows:—

| Chemicals Consumed. | Weight. | | Value. | | | |
|---|---|---|---|---|---|---|
| | Pounds per 2000 lbs. of Ore. | Per cent. of Ore. | Average Cost. | | | |
| | | | Per £20 Value of Bullion obtained. | | Per 2000 lbs. of Ore Stamped. | |
| | | | s. | d. | s. | d. |
| Quicksilver . . . | 1·39 | 0·07 | 10 | 3 | 2 | 7 |
| Salt, at 94s. per ton . | 45·29 | 2·26 | 10 | 4 | 2 | 7 |
| Sulphate of copper . | 2·09 | 0·10 | 2 | 8 | 0 | 8½ |
| Sulphuric acid . . | 0·56 | 0·03 | 0 | 6½ | 0 | 1½ |
| Total value of chemicals consumed . . | | | 23 | 9½ | 6 | 0 |
| Firewood, at 17s. 9d. per cord . . . . . | | | 15 | 6 | 3 | 10 |
| Other stores . . . . . . . . . . | | | 18 | 3 | 4 | 7 |
| Pay roll, at 14s. per day per man . . . . | | | 34 | 6 | 8 | 7 |
| Salaries . . . . . . . . . . | | | 3 | 10 | 0 | 11½ |
| General charges . . . . . . . . . | | | 0 | 8½ | 0 | 2 |
| Sundries . . . . . . . . . . | | | 0 | 11 | 0 | 3 |
| | | | 97 | 6 | 24 | 4½ |

## CHLORINATION PROCESS.

This method, originally introduced by the late Professor C. F. Plattner of Freiberg for the extraction of a minute quantity of gold from the arsenical pyrites of Reichenstein in Silesia, is based on the fact that chlorine gas transforms gold into auric chloride, which is soluble in water, without materially attacking the metallic oxides with which it may be associated. From the chloride solution thus obtained, gold may be precipitated, either in the metallic form, by iron, copper, or a solution of ferrous sulphate; or as sulphide by sulphuretted hydrogen. No metallic sulphides or arsenides must, however, be present, as these substances would be transformed into chlorides, causing an unnecessary expenditure of chlorine. The presence of sulphur is also injurious from giving rise to the production of chloride of sulphur, which, in the presence of water, becomes transformed into hydrochloric and sulphurous acids, when a certain amount of the metallic oxides is dissolved. The whole of the sulphur must, therefore, be expelled by careful roasting, while the iron is, at the same time, converted into peroxide. The chlorine employed should also be free from hydrochloric acid.

In California, the extraction of gold from auriferous pyrites by the chlorination process is conducted in the following manner:—

The concentrated tailings are roasted in a reverberatory furnace, until no further smell of sulphur is perceptible, a little charcoal being sometimes introduced towards the close of the operation for the purpose of decomposing any sulphates or arsenical salts that may have been produced. This roasting is effected at a low temperature in order to avoid agglomeration of the ore, and, at the expiration of from six to eight hours, the charge is withdrawn and spread on the floor to cool. When sufficiently cold it is repeatedly turned and sprinkled with water, so that it may become regularly moistened throughout. The success of the subsequent operation depends, in no small degree, on the amount of water thus added, and the uniformity of its mixture with the ore. After having been properly moistened, the roasted pyrites is charged into large wooden tubs, 7 feet in diameter and 2 feet 6 inches in depth. These are provided with perforated false bottoms, beneath which the chlorine is introduced, and thence ascends through the damp and finely divided auriferous ferric oxide, which ultimately becomes permeated by the gas. The chlorine is produced from a mixture of common salt, peroxide of manganese, and sulphuric acid, contained in a leaden generator, which communicates with the space beneath the false bottom by means of a lead pipe. There is also a plug-hole in the bottom of each tub for the purpose of draining off the auriferous solution obtained. After being charged with moistened ore each tub is closely covered by a wooden lid, and chlorine is introduced beneath the false bottom. At the expiration of some hours, the whole mass has become strongly penetrated by chlorine, which, as a greenish gas, lies heavily above the tailings. In this condition the tub and its contents are allowed to remain from ten to fifteen hours, at the expiration of which period, the cover is removed and clean water introduced. As soon as the water has risen to the surface of the charge, the plug-hole at the bottom is opened, and the water containing the dissolved chloride of gold is run off into glass carboys. The gold is subsequently precipitated in the metallic form by the addition of ferrous sulphate, and forms a brownish-black deposit on the bottom of the carboys; this reaction is expressed by the following equation:

$$2AuCl_3 + 6FeSO_4 = Au_2 + Fe_2Cl_6 + 2Fe_2(SO_4)_3.$$

The precipitated gold thus obtained is collected on filters, dried, and afterwards fused with borax in black-lead crucibles. The ingots prepared in this way are usually 995 fine. When the gold is in a finely divided state this process affords satisfactory results; but the larger particles of metal not being completely dissolved in the time necessary for the solution of the smaller ones, a loss must necessarily result unless the time during which the ores are exposed to the action of chlorine is sufficiently prolonged to effect the solution of the largest grains of gold present.

It follows that the chlorination process is most advantageously applied to ores in which the precious metal is uniformly and finely divided. Any silver that may have been present in the original sulphides, as well as that constantly alloyed with the gold, is by this process converted into chloride of silver, which is insoluble in the auriferous solution, and therefore remains with the residues in the tubs.

Mr. H. R. Cassel proposes to obtain chlorine for the treatment of gold ores by the electrolysis of common salt in the presence of the ore instead of preparing it in a separate generator.

### Methods for Smelting Gold Ores.

Auriferous minerals unsuited for amalgamation are generally smelted with iron pyrites, copper, or lead ores, the gold being ultimately concentrated in metallic copper or silver, from which it is finally separated by one of the methods of parting described subsequently. In other respects the order of the operation is the same as in smelting ordinary copper, lead, or silver ores.

Gold alloys readily with iron, a property that was formerly utilized to a small extent in the treatment of black sand obtained in the alluvial gold washings of the Ural. This consists chiefly of magnetite, and when smelted with the necessary fluxes and charcoal gave an auriferous cast-iron, from which the gold was recovered by dissolving it in sulphuric acid.

In Parkes's process of desilverizing, copper and gold, if present in lead, are almost entirely removed by the zinc first added; so that if the first zinc skimmings are treated apart, a minute quantity of gold may be sufficiently concentrated in the silver obtained to render the latter worth parting.

At Offenbanya and Zalathna in Transylvania, auriferous pyrites, containing a little copper and from 11 to 24 ounces of auriferous silver per ton, has since 1876 been treated by the following combination of smelting and extracting processes:—

1. The ores, partly in lumps and partly in a finely divided state, are roasted, the sulphur gases being collected and converted into sulphuric acid.

2. The roasted ores are smelted with quartz flux for coarse metal, which is essentially ferrous sulphide, containing sulphur 28·7, iron 69·6, and copper 0·6 per cent., and is ground to a fine powder.

3. The pulverized coarse metal is digested with dilute sulphuric acid in closed wooden vessels lined with lead for twelve hours, when about half the iron is converted into ferrous sulphate, while the insoluble residue, amounting to about one-fourth of the charge, is proportionately

enriched, and contains $3\frac{1}{2}$ per cent. of copper and 60 to 80 ounces of gold and silver per ton. The acid liquor is treated for ferrous sulphate, and the sulphuretted hydrogen is mixed with sulphur-dioxide in order to recover the sulphur.

4. The extracted residues are smelted with various additions, such as rich ores, raw regulus, litharge, and other lead products, in a small blast-furnace, producing lead with about 40 ounces of auriferous silver per ton and some regulus. The lead is refined in the usual way, and the regulus is subjected to a second extraction with sulphuric acid, and a series of concentrating and desilverizing fusions, until it is enriched to about 25 per cent. of copper.

5. The copper regulus is digested in a cast-iron pot with heated strong sulphuric acid, leaving a residue consisting chiefly of lead and copper, with some silver and gold, which is returned to the lead-furnaces. The acid liquor is run over copper-plates to decompose silver sulphate, and when clarified is either crystallized for blue vitriol or treated with iron to obtain cement copper.

Extraction of Silver and Gold from Copper.—Small quantities of gold and silver may be separated from copper by means of sulphuric acid, cupric sulphate being produced while the precious metals remain in the insoluble residues; but for the success of the operation it is necessary to use the acid in a diluted state, as when strong it has no action upon copper. It is also necessary to oxidize the metal either by a preliminary roasting or by moist air, so that the acid may be entirely used in dissolving cupric oxide and not wasted in the formation of sulphur dioxide. The first method was formerly practised in Colorado in the treatment of auriferous copper bottoms (p. 732),[1] which were liquated to remove lead and then refined by oxidation until the whole of the sulphur was removed, when it was granulated by casting it into cold water. The granules were afterwards exposed in a calciner to a strong red heat for thirty-six hours, and became entirely converted into a mixture of cupric and cuprous oxides, which was powdered and dissolved in weak sulphuric acid. The residues, when cleared from the vitriol liquors and melted in plumbago crucibles, gave a mixed bullion of 600 to 800 fine, containing about two parts of gold to one of silver, which was sent to the Mint refinery.

The second method of dissolving copper, by the joint action of air and sulphuric acid, known as *vitriolizing*, is used at Oker, in the Lower Harz, where it was introduced in 1858 as a substitute for liquation in the treatment of copper containing gold and silver that could not be refined to advantage.

The following notice of this process[2] is condensed from the description contributed to the preceding edition of this work by Professor F. Ulrich, formerly of Oker but now of Hanover.

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[1] 'Proceedings of the American Institute of Mining Engineers,' vol. iv. p. 297.

[2] For fuller details see 'Zeitschrift für Berg- Hütten- u. Salinenwesen,' vol. xxv. p. 163.

The copper produced at Oker is of two kinds: one being of good quality and poor in silver is refined and sold as rosette copper; while the other, which contains, in addition to gold and silver, various impurities prejudicial to its use as metal, is converted into vitriol, and the precious metals are removed from the insoluble residues in the following manner:—

The metal, in the state of black copper containing about 50 ounces of silver per ton, is granulated and placed in large wooden tubs lined with thick sheet-lead. On the perforated false bottom of the tub is first laid a layer, 6 inches in thickness, of large fragments of copper, and upon this is placed the granulated metal to a depth of 3 feet 6 inches, which is then moistened with warm dilute sulphuric acid and exposed to the air. Oxidation of the copper rapidly takes place, and the granules become covered with a dark film. A fresh supply of warm dilute sulphuric acid is now added, by which the sulphate formed is dissolved. From a hole in the bottom of the tub the acid solution of sulphate of copper runs into a slightly-inclined lead-lined gutter, about 160 feet long, where it becomes sufficiently cooled to deposit nearly the whole of the dissolved sulphate in small irregular crystals. The mother liquors, which are nearly free from copper but contain a large amount of sulphuric acid, are collected in a closed reservoir, and blown by an injector into a lead-lined cistern, where they are re-heated by steam, and returned by a syphon to the tubs containing the granulated copper as required. The density of the liquors is 30° Baumé, and they are heated to from 87° to 100° C. Each time the liquors are run off, the copper is exposed during fifteen minutes, for the purpose of oxidation, and sulphuric acid is run on until the copper has regained its bright metallic appearance.

When the crystallized deposit in the gutter has attained a thickness of about 3 inches, it is removed by copper chisels and thrown upon an inclined plane, from which the adhering liquors drain back into the gutter. This crude sulphate of copper is taken to leaden boiling pans, 12 feet in length, 11 feet in width, and 2½ feet in depth, where it is dissolved, either in water or in weak acid liquors. The quantity of liquid is so regulated that a solution is obtained which, at 87° C., has a density of 30° Baumé.

This solution is allowed to settle during twelve hours, care being taken to avoid, as far as possible, any decrease of temperature. At the expiration of this time the liquor, which is perfectly clear, is run into crystallizing pans, 10 feet long, 5 feet wide, and 4 feet deep, in which are hung numerous strips of lead. On these, in the course of from twelve to fourteen days, are formed large crystals of sulphate of copper, which are subsequently removed and packed for the market.

The mud, containing the silver and gold, which settles in the boiling pans, is taken out and mixed with litharge. This mixture is smelted for lead, and the quantity of litharge is so regulated that the resulting alloy shall contain about 2 per cent. of silver. The lead thus obtained is passed to the refinery, and affords silver containing from 1·5 to 1·7 per cent. of gold.

The loss of lead is 8·4 per cent., and the silver obtained slightly exceeds the amount indicated by assay.

One part of granulated copper requires 2·4 parts its weight of acid of 30° Baumé to dissolve it, and produces 3·8 parts of sulphate. A set of six dissolving tubs, two pans, and twenty-four crystallizers is required to produce 30 cwts. of crystals of blue vitriol in twenty-four hours.

### Parting of Gold from Silver.

The separation of gold from silver on the larger scale is now generally effected by sulphuric acid, which is cheaper than nitric acid, and does not require expensive platinum or porcelain vessels, the operation being performed in cast-iron pots. In order that the alloy may be completely attacked, it should not contain more than 35 per cent. of gold, and from the slight solubility of sulphate of copper in strong sulphuric acid, it is of importance that it should not contain beyond 10 per cent. of copper.

The alloy, after the additions necessary to bring it to the proper standard have been made, is melted either in large crucibles or in a reverberatory furnace, and granulated by being poured into water. The granulated metal, in charges of 2 to 5 cwts., is placed, with 2½ times its weight of sulphuric acid of sp. gr. 1·840, in cast-iron pots, which are heated to ebullition by a fire beneath. The sulphuric acid under these circumstances acts rapidly on the metal, sulphate of silver is formed, and sulphurous anhydride evolved; this is conducted by a leaden dome, connected with a well drawing chimney, placed over each pot during the time the attack is being made, into a chamber, where it is reconverted into sulphuric acid. At the expiration of four hours the attack is completed, and a certain quantity of sulphuric acid of the sp. gr. 1·69, obtained by the concentration of the acid mother liquors from the crystallization of sulphate of copper, is added, and boiled during a few minutes, when the fire is withdrawn from beneath the pots, and the liquors are diluted and allowed to stand, in order that the finely divided gold may be deposited on the bottom. When the supernatant liquor has become clear, it is drawn off by a syphon into lead-lined evaporators, heated by steam-pipes, and partially filled with mother liquors remaining from the crystallization of sulphate of copper. The temperature is then raised until the whole of the sulphate of silver, which began to fall on cooling, is re-dissolved, and a further deposit of gold is obtained. The liquor is next syphoned into other evaporators, in which copper bars are suspended, where the silver is rapidly precipitated in the form of a crystalline powder. In the course of a few hours the last traces of silver sulphate are completely decomposed, and the metallic deposit, after being carefully washed, is compacted by hydraulic pressure into rectangular bricks, which are fused in large earthen crucibles, and cast into ingots. The silver thus obtained contains from 3 to 5 thousandths of copper.

The pulverulent gold obtained by the first attack still contains silver, and is therefore again subjected to the action of strong sulphuric acid, in smaller pots, heated from beneath. These are made usually of cast-iron, but sometimes of platinum.

It is then washed, or "sweetened," on a filter with hot water, dried, pressed, and melted in a black-lead crucible; borax and nitre being sometimes used to remove the last traces of impurities. Under favourable conditions the resulting bar of gold may be about 996–998 fine.

The solution of copper sulphate produced during the precipitation of silver by copper bars is evaporated in a shallow cistern lined with lead, and heated by a series of steam-pipes laid in zigzag across the bottom. When the liquors have in this way been sufficiently concentrated, they are syphoned off into large tubs lined with lead and bound with copper or wooden hoops, as, from the readiness with which sulphate of copper acts on iron, bands of this metal would be rapidly attacked by the liquor accidentally spilt over the sides of the vessels. After having been filled, these tubs are closely covered to prevent their too rapid cooling, and, after the expiration of about ten days, the mother liquors are drawn off, and the crystals of sulphate of copper adhering to the sides carefully removed. These mother liquors, when again concentrated, yield a further supply of crystallized salt, after which they are set aside, to be employed in place of sulphuric acid, as already described.

When sulphate of copper of superior purity is required, the crystals first obtained are sometimes subjected to a second crystallization, but in the majority of cases they are merely washed on a wicker sieve, and after being allowed to drain in a large leaden cullender, are packed in strong casks for the market.

From the economy with which this process is conducted, and the comparatively low price of sulphuric acid, it sometimes admits of being advantageously applied to the refining of silver containing 0·0005 only of gold.

At Oker gold is refined by sulphuric acid in porcelain vessels, which are protected with a wire network and clay covering, and are heated in sand baths. They are cleanly in use, but require very careful handling, and are only suited for small charges (28 lbs. at one time). The gold obtained is 985 fine.

The parting of gold and silver may be effected by the use of nitric acid, but when the resulting nitrate of silver cannot be advantageously utilized, it must be converted into chloride before it can be reduced by metallic copper.

In the United States' Mint refineries so-called double-parting processes are used. The bullion, which is so mixed as to contain 100/285 of gold, is granulated and heated first with nitric acid in glazed earthenware jars, which removes the silver to within 6 per cent., and subsequently is twice boiled with sulphuric acid in three-legged cast-iron pots, giving a gold of 998–999 fine. The silver salt is converted into chloride and reduced by granulated zinc.

In America, it is also customary to dissolve doré silver in bars without previous granulation, which is found not only to facilitate the regular action of the acid, but also to give a coarser gold that is more easily washed than that from granulated metal. The reduction of sulphate of silver may also be effected by scrap iron or ferrous sulphate; the former reagent is used at Frankfort and Lautenthal, and the latter at San Francisco. By adding scrap iron to the liquors, in the second case, the ferric sulphate contained is reduced to ferrous sulphate, and may be used for precipitating fresh quantities of silver.

## PREPARATION OF PURE GOLD.

Gold of extremely high fineness cannot be obtained by sulphuric acid parting alone. When the washed gold from the second boiling is heated with bisulphate of sodium, silver may be removed to within 2 or 3 thousandths, but platinum when present is not sensibly attacked by sulphuric acid, although it may be partially removed if nitric acid is used at first, or if the gold from the bisulphate treatment is fused with nitrate of sodium or potassium. When gold is required of absolute purity, or as near an approach to it as can be got, as for example for use in check assaying, a quantity of assay cornets or other kinds of parted gold is dissolved in aqua regia, and after removal of excess of acid, potassium chloride and alcohol are added to precipitate platinum. The auric chloride is dissolved in a very large quantity of water and left at rest for a considerable time, that the chloride of silver existing in the form of suspended particles may subside. The clear liquor is then syphoned off, and the gold is precipitated by a solution of ferrous sulphate or oxalic acid, or by a current of gaseous sulphur dioxide. The last of these reagents is probably the safest, when it is desired to prevent the possibility of contamination by foreign metals, but liquid precipitants are more convenient. Mr. Roberts-Austen, of the Royal Mint, has in this way obtained gold of the apparent fineness of 999·96 thousandths, using oxalic acid for precipitation. Platiniferous gold may also be refined if it is cast into a plate and used as the anode in an electrolyzing cell, having for a cathode a plate of the purest gold that can be got. The electrolyte is a neutral solution of auric chloride. As the anode dissolves under the action of the current, platinum and the allied metals are set free and are collected at the bottom as a black powder. As the gold deposited on the cathode is exactly equivalent to that dissolved from the anode during the same time, the strength of the solution remains unchanged during the operation.

## REFINING BY CHLORINE GAS.

This process was invented by Mr. F. B. Miller, assayer in the Sydney Branch of the Royal Mint, and is applied to the treatment of gold bars when the proportion of silver present is not materially in excess of 10 per cent. It consists in passing a current of chlorine gas through the gold *while in a melted state*, whereby chloride of silver is formed, which, being of low specific gravity, rises to the surface of the melted gold, while the latter remains in a purified condition beneath.

Chloride of silver has always been considered a somewhat volatile substance, but, in practice, it is found that its volatility is not so great as might have been anticipated, and that, if its surface is coated with a layer of fused borax, it may be kept melted at a high temperature without material loss.

The furnace required for the operation is an ordinary 12-inch square gold-melting furnace, having the flue as near the top as possible, so as to allow of the crucible standing high up in it without being cooled by the draught, and of such a depth that the bottom of the pot, when it is placed in the fire, may not be more than 3 inches above the bars.

The covering of the furnace should consist of two fire-tiles, 7½ inches wide and 15 long, one of which should have a slot or hole in its centre, for the clay chlorine-pipes to pass through. An iron cover will not answer, as it becomes much too hot for convenient working.

The crucibles in which the refining is performed should be French white fluxing-pots; ordinary black-lead pots will not answer, owing to the reducing action they exert on the compounds formed. To prevent infiltration of the very liquid chloride of silver into the pores of the pots, they are prepared by filling them with a boiling saturated solution of borax in water, which is allowed to stand for ten minutes, and then poured off, the crucibles being afterwards set aside to dry. The borax forms a glaze on the inner surface of the crucibles when they become hot in the furnace.

When used for refining, these French crucibles are placed within black-lead pots, as a precaution against loss, should the former crack, which, however, seldom happens. The crucibles are covered with loosely fitting lids, with the requisite holes through them for the passage of the clay chlorine-pipes, &c. A pipe, ½ inch in diameter, 22 inches long, and of $\frac{3}{16}$-inch bore, has been found to answer all requirements. The chlorine generators should consist of the best glazed stoneware acid-jars, each capable of holding from 10 to 15 gallons, and furnished with two necks. One of these openings should be stopped with a vulcanized indiarubber plug, through which should pass tightly two glass tubes, the eduction-tube and the safety- or pressure-tube; the length of the former being a few inches, and of the latter 8 or 10 feet. The other opening, intended for introducing oxide of manganese, &c., should be closed with a leaden plug, covered with a stout piece of indiarubber, and well secured.

Each generator should be charged with a layer of small quartz pebbles, down nearly to the bottom of which the pressure-tube should extend. On this layer should be placed from 70 to 100 lbs. of binoxide of manganese, in grains about ¼-inch cube, freed from powder by sifting. This quantity will be sufficient for many refining operations, and will obviate the necessity of repeated dismantling of the apparatus.

Each generator should be suspended to about half its height in a water-bath of galvanized-iron. The chlorine gas is produced, when required, by pouring common hydrochloric acid down the safety-tube, the apparatus being warmed by means of gas-burners beneath the water-baths. The gas is conveyed from the generators by a leaden pipe fitted with branches to supply the several furnaces, all intermediate connections being formed by means of vulcanized india-rubber tubing, which, if screened from direct radiation from the fire, stands the heat well, even immediately over the furnaces.

Screw compression-clamps on the india-rubber tubes give the means of regulating the supply of gas as required, and enable the operator to shut it off entirely as soon as the refining is over. The chlorine then, having no means of escape, accumulates in the generator, and soon forces all the acid up the safety-tube into a vessel placed above to receive it, and, the acid no longer acting on the oxide of manganese, the supply of gas ceases.

Two such generators, and three ordinary gold-melting furnaces, are capable of refining 2,000 ounces of gold, containing about 10 per cent. of silver, between 9 A.M. and 2 P.M.

As soon as the gold is melted, from 2 to 3 ounces of borax in a state of fusion are poured upon its surface. If the borax is added sooner, it acts too much on the pot; and, if thrown in cold, is liable to chill the gold. The clay pipe which is to convey the chlorine to the bottom of the melted gold is now introduced. At the moment of its entering the melted gold, the screw compression-clamp is slightly loosened, so as to allow a small quantity of gas to pass through it, and thus prevent any metal rising and setting in the pipe, which is then gradually lowered to the bottom of the molten gold, where it is kept by means of weights attached to the top. The compression-tap is now relaxed, and the gas is heard bubbling through the melted metal, sufficient hydrochloric acid being, from time to time, added to the generators to keep up a rapid evolution of chlorine.

The column of liquid in the safety-tube, acting, as it does, like a barometer, affords a ready means of knowing the pressure in the generator, and of judging of the rate of production of the gas, as well as at once showing, by its fall, if anything irregular has occurred—such as a leak or a crack in the chlorine-pipe or pot. From 16 to 18 inches in the safety-tube correspond to and balance 1 inch of gold in the refining-crucible. When the chlorine is first introduced into the melted gold, fumes are seen to pass up from the holes in the crucible lid; these are not chloride of silver, but volatile chlorides of some of the baser metals. They are especially dense when much lead is present in the alloy under treatment, forming a white deposit on any cold substance presented to them. After a time, longer or shorter, according to the nature of the impurities in the gold, these fumes cease. So long as any decided quantity of silver is present in the molten gold, the whole, or nearly the whole, of the chlorine is absorbed; little, if any, appearing to escape.

When the refining is nearly complete, fumes of a darker colour than those first observed make their appearance, and the end of the operation is indicated by a peculiar flame or luminous vapour of a brownish-yellow colour, occasioned by the escape of free chlorine. This, however, is not a sufficient indication: the process is not finished until this flame imparts to a piece of white tobacco-pipe, when held in it for a moment, a peculiar reddish or brownish-yellow stain.

When these appearances are observed, which usually happens in about an hour and a half after the introduction of chlorine, the gas is shut off, and the pots are removed from the fire; the white crucible is lifted out of the black one, and, together with its contents, is allowed to stand until the gold becomes cold enough to solidify. The chloride of silver, which remains liquid much longer, is then poured off into iron moulds. The crucible is now inverted on an iron table, when the still red hot gold falls out in the shape of a cone; this is slightly scraped, and then thrown into a concentrated solution of common salt, to free it from any adherent chloride of silver.

An alloy containing originally 89 per cent. of gold, 10 per cent. of silver, and 1 per cent. of base metals, will yield, on an average, a cake of chloride weighing, with a little adherent borax, 16 ounces for every 100 ounces operated on.

The gold is now fine, and simply requires casting into ingots.

As before stated, it is found that all these operations can readily be performed, and 2,000 ounces refined per day in three common melting furnaces, in about five hours; 98 per cent. of the gold originally contained in the alloy operated on is then ready for delivery.

The other 2 per cent. remains with the chloride of silver, partly in the metallic state, and partly in a state of combination with chlorine, and probably with silver.

To free the chloride of silver from this combined gold (that mechanically mixed being eliminated at the same time), it is melted in a boraxed white pot, with the addition of from 8 to 10 per cent. of metallic silver, rolled to about $\frac{1}{8}$ inch thick. The chloride of gold is by this means reduced at the expense of the metallic silver, chloride of silver being formed; while the liberated gold sinks, and together with the excess of silver, melts into a button at the bottom of the pot. As soon as the whole is thoroughly melted, the pot is removed from the furnace, and allowed to stand about ten minutes. The still liquid chloride of silver is then poured into large iron moulds, so as to form slabs of a convenient thickness for the next operation, namely, its reduction to the metallic state, by strips of silver. This is done by attaching the slabs to a frame, which is lowered into a vat containing another frame with zinc plates, so arranged that the plates and slabs alternate with each other, and form a zinc and chloride of silver battery. Water is used as the exciting liquid; and when the two elements are brought into connection by strips of silver attached to the frames, an energetic galvanic action is set up, and in about 24 hours the slab of chloride, while retaining its shape, is found to be converted into spongy metallic silver.

The fineness of the gold reduced by this process varies from 991 to 997 in 1,000 parts, the average being 993·5; the remaining 6½ thousandths are silver. This compares favourably with any of the previously known practical processes, none of which leave less silver in the resulting fine gold.

The silver resulting from this method of refining is tough, but its quality varies somewhat, according to the gold originally operated on; if the alloy treated contains much copper, the greater part of this remains with the resulting silver, but the other metals are nearly all eliminated.

The fineness of the silver hitherto obtained has varied from 918·2 to 992·0 in 1,000 parts, the average being 965·6.

An analysis of the silver resulting from refining gold, known originally to have contained, among the base metals in the alloy, copper, lead, antimony, arsenic, and iron, gave the following results:—

| | |
|---|---|
| Ag | 972·3 |
| Cu | 25·0 |
| Au | 2·7 |
| Zn and Fe | traces |
| | 1,000·0 |

Miller's process has been successfully used in the Royal Mint by Mr. Roberts-Austen for toughening brittle gold.

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## PLATINUM.

In its pure state, particularly when refined by the process of Deville and Debray, platinum is nearly as white as silver, is capable of receiving a high polish, and is very ductile and malleable. Platinum is softer than silver, but its hardness is much increased by the presence of even a minute quantity of iridium. It resists the strongest heat of a wind-furnace, but may be fused by the electric current or by the oxyhydrogen blowpipe, before which it is dispersed with scintillation. The melting point is 1775° C.; at 1600° C. it may be welded like iron. According to Deville and Debray, it absorbs oxygen in the fused state, and when melted in considerable masses, on cooling, spirts like silver. Platinum is one of the heaviest of all known substances, having the specific gravity of 21·46, which is only exceeded by osmium (22·47) and iridium (22·40). The thermal and electric conductivities are 8·4 and 16·4 respectively (silver = 100). Platinum is not oxidized by the air at any temperature, and is not attacked by any single acid; by aqua regia it is dissolved with formation of platinic chloride, $PtCl_4$. It is also attacked, at a red heat, by acid potassium sulphate, by the caustic alkalies and by the alkaline earths, especially by the hydrates of lithium and barium, but is not affected by alkaline carbonates, even when exposed to their action at very high temperatures. A mixture of nitre and caustic potash acts with greater rapidity than the alkali alone; and platinum foil, when heated in presence of arsenic, sulphur, or phosphorus, loses its malleability and ductility. When these bodies are brought, at a high temperature, in contact with platinum in a state of fine division, combination takes place, and brittle fusible compounds result. A mixture of silica and charcoal attacks platinum at high temperatures, producing silicide of that metal, and for this reason platinum crucibles, which have been frequently ignited in an open fire, lose their flexibility, and become rough on the outside.

Platinum alloyed with a large quantity of silver is soluble in nitric acid, and consequently small quantities of this metal may be removed from gold by inquartation with silver, and parting with nitric acid in the ordinary method of gold assaying.

Platinum possesses the remarkable property of condensing gases upon its surface, and determining their combination. This property, exhibited even by clean plates of platinum, and in a greater degree by the spongy metal, obtained by the ignition of ammonio-platinic chloride, is most marked in the extremely divided form known as *platinum-black*.

Platinum-black is prepared in various ways. A common method is by boiling a solution of platinic chloride, $PtCl_4$, with carbonate of sodium and sugar. Chloride of sodium is formed, a portion of the sugar is decomposed with evolution of carbonic anhydride, and platinum is precipitated in the metallic state. It may also be made by dissolving platinous chloride, $PtCl_2$, in boiling caustic potash, and gradually adding alcohol to the solution. Rapid evolution of carbonic anhydride takes place, and the metal is precipitated in a state of extreme division. The same result may be obtained by decomposing platinum sulphate by heat and strong alcohol. Platinum-black prepared by any of the foregoing processes, when dried, resembles lamp-black, and soils the fingers in the same way; it may be heated to full redness without any change of its appearance or properties, but at a white heat it assumes a metallic aspect. It is sometimes employed for eudiometrical experiments.

DISTRIBUTION OF PLATINUM.—This metal is found in a native state, and occurs, alloyed with various others, in alluvial deposits similar to those from which gold, its frequent associate, is obtained. Sands producing platinum are found principally in valleys traversing serpentine. Native platinum generally presents the appearance of small grains, of a greyish-white colour, approaching to that of tarnished steel. These grains are commonly flattened, and appear to have been polished by friction against other bodies. Their size usually varies from that of linseed to that of hempseed, but fragments of much larger dimensions have occasionally been discovered. One piece brought from Choco, New Granada, by Humboldt, and presented to the Berlin Museum, weighs 1,088 grains, or above two ounces avoirdupois. The Madrid Museum possesses a specimen found in 1822, in South America, which is as large as a turkey's egg, and weighs 11,641 grains. A specimen of this metal was found, in the year 1827, on the eastern slope of the Urals, which weighed 11·57 lbs. troy. The largest specimen yet discovered weighs 21 lbs. troy, and is in the cabinet of Count Demidoff.

The substance known as native platinum is not a pure form of that metal, being usually combined with osmium, iridium, palladium, rhodium, and ruthenium, forming the so-called *Polyxene* group of elements, besides gold, silver, iron, and copper. It is also frequently associated with other minerals, such as osmiridium, gold, magnetite, ilmenite, chromic iron ore, and iron pyrites.

Four specimens of native platinum afforded, on analysis, the following results:—

| | 1. | 2. | 3. | 4. |
|---|---|---|---|---|
| Pt | 80·87 | 82·60 | 85·50 | 80·00 |
| Au | ... | 0·20 | 0·80 | 1·50 |
| Fe | 10·92 | 10·67 | 6·75 | 7·20 |
| Ir | 0·06 | 0·66 | 1·05 | 1·55 |
| Rh | 4·44 | ... | 1·00 | 2·50 |
| Pd | 1·30 | ... | 0·60 | 1·00 |
| Cu | 2·30 | 0·13 | 1·40 | 0·65 |
| Os | ... | ... | ... | ... |
| Iridosmine | 0·11 | 3·80 | 1·10 | 1·40 |
| Sand | ... | ... | 2·95 | 4·35 |
| | 100·00 | 98·06 | 101·15 | 100·15 |

1. Ural, by Osann; 2. Borneo, by Böcking; 3. California, by Deville and Debray; 4. Choco, by Deville and Debray.

Platinum was discovered (1735) by Ulloa, a Spanish traveller, in the alluvial deposits of the river Pinto, in the district of Choco, New Granada. It has since been found in the Ural Mountains, in the Island of Borneo, in the sands of the Rhine, in those of the Jacky in St. Domingo, and in the gold regions of Brazil, California, &c. The grains of gold are separated from platinum by amalgamation. The grains of platinum which are found in the sands of the river Jacky, in St.

Domingo, are extremely brilliant, and are intermixed with a siliceous sand, which is generally ferruginous. The largest proportion of the platinum at present produced is obtained in the Ural districts from the auriferous sands of Miask, Nijne-Taguilsk, Goroblagodatsk, &c.

Russia annually affords about 4,400 lbs. troy of this metal, which is about five times the amount of the united products of Brazil, Borneo, St. Domingo, and the United States of Colombia.

ESTIMATION OF PLATINUM.—This metal, for the purposes of analysis, is weighed either in the metallic state, or in the form of ammonio-platinic chloride, $2NH_4Cl,PtCl_4$, which is collected on a tared filter, and dried at a temperature of 100° C.

When platinum is contained in a solution in the form of chloride, the liquor is first concentrated by evaporation, and subsequently mixed with about twice its volume of alcohol. Solution of chloride of ammonium is now added in excess, and the liquid again concentrated by evaporation in a water-bath. By this means ammonio-platinic chloride is precipitated, and after being carefully washed, first with dilute solution of sal-ammoniac, and afterwards with a mixture of alcohol and ether, is dried in a water-bath. From the weight of the double salt obtained, the percentage of platinum is readily deduced, as every 100 parts of the former correspond to 44·20 parts of metallic platinum. Instead of deducing by calculation the weight of platinum from that of the double salt obtained, its amount may be at once determined by decomposing the ammonio-salt by ignition, and weighing the metallic spongy platinum which remains.

For this purpose the double chloride should be exposed to a full red heat in a closed porcelain crucible, protected from direct action of the fire by being enclosed in one of fire-clay. The decomposition of this salt may likewise be effected in a gas-furnace, in which case the external crucible may be dispensed with. This decomposition of the salt by heat requires to be conducted with great care, since if the evolution of ammonium chloride be too rapid, a notable amount of metallic platinum will be carried off. Chloride of potassium may be used instead of chloride of ammonium for the precipitation; the potassio-platinic salt, $2KCl,PtCl_4$, produced, containing 40·4 per cent. of platinum, is either dried and weighed at 100° C., or is decomposed, by heating to redness, into metallic platinum, and potassium chloride. The latter is separated by solution in hot water, and the former dried and weighed in the metallic state.

Native platinum cannot be assayed in the dry way, and its complete analysis is a long and difficult operation, which can only be successfully undertaken by an experienced chemist.[1] The commercial assay of platinum is conducted by performing, on a small scale, one of the processes now to be described.

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[1] The first somewhat complete examination of native platinum was made by Wollaston; but both Berzelius and Vauquelin added much to our knowledge of the chemistry of this metal. The processes employed by the former for the analysis of native platinum are described in Gmelin's 'Handbook,' vol. vi. p. 259. Claus, in 1854 (*Beiträge zur Geschichte der Platinmetalle*), proposed a simpler and in some respects more accurate method. Another method has been devised by Deville and Debray (Ann. Ch. Phys. (3) lvi. p. 385). Descriptions of the two processes last referred to are given in Watts's 'Dictionary of Chemistry,' Art. 'Platinum.'

## METALLURGY OF PLATINUM.

Wollaston's Process.—The platiniferous grains subjected to treatment, besides containing the metal principally sought, also yield variable quantities of osmium, iridium, rhodium, and ruthenium; they also frequently contain, in addition to these, gold, silver, iron, and copper, together with various heavy minerals, such as titaniferous and chrome iron ores.

When gold is present in sufficient quantity, the ore is first subjected to amalgamation for the purpose of its extraction, and the residue, after careful mechanical washing and digestion, first with nitric acid and subsequently with hydrochloric acid, is treated for platinum. The concentrated ore is attacked by aqua regia, containing an excess of hydrochloric acid, either in large glass carboys or in stoneware vessels, heated on a sand-bath placed under a chimney, by which the evolved fumes are carried off. The aqua regia is diluted with water, as by this means a smaller quantity of iridium is dissolved than when the acids are employed in an undiluted state, and when this metal is present even in small quantity the manufactured platinum is rendered hard and its tenacity impaired. The aqua regia is several times renewed before the solution of the ore is completed, and care is taken to avoid the inhalation of the escaping fumes, which, from the presence of osmium compounds, are extremely prejudicial. The solution thus obtained is set aside, in order that it may brighten by subsidence, and the clear liquid, after being drawn off by a glass syphon, is treated with solution of sal-ammoniac as long as a yellow precipitate is deposited. The mother liquors from this precipitate still contain a considerable amount of platinum, together with variable quantities of the other metals originally present in the material operated upon. Bars of zinc, when introduced into the liquors, produce a deposit of a dark colour, from which a certain amount of platinum is obtained by washing it clean with hot water, and subsequently attacking it by aqua regia containing a large excess of hydrochloric acid, to prevent the precipitation of palladium or lead if contained in the solution. Sal-ammoniac is now added to the clear solution, and a second precipitate of ammonio-platinic chloride is obtained.

The double chloride thus obtained is heated to redness in large black-lead crucibles, and by this means chlorine and sal-ammoniac are expelled, while metallic platinum, in a spongy state, remains.

This spongy platinum is next finely pulverized, by being rubbed between the hands, and afterwards intimately mixed with water, so as to form a dense black slime. This is carefully passed through sieves of fine wire-gauze, and the coarser particles, which remain on the meshes, are again crushed and ultimately made to pass through.

In conducting this operation, it is of importance to avoid the use of any hard body, by which a commencement of aggregation between the particles of metal might be produced. Scrupulous cleanliness on the part of the workmen is also necessary, to prevent the introduction of any extraneous matter into the finely divided mass, which might be sufficient to cause a serious imperfection in the forged platinum produced. To avoid this, the metallic powder is repeatedly washed, by decantation, previous to its consolidation.

The platinum paste is next moulded in an apparatus consisting of a gun-metal cylinder, accurately fitted with a steel piston, and enclosed at the lower end in a steel foot-piece, by which the escape of the pasty mass is prevented. Care is taken that the mass to be compressed be entirely free from air-bubbles, and, after first ramming with a wooden pestle, the steel piston is applied. The water is thus separated from the metallic particles, and their closer compression is afterwards effected by a hydraulic or powerful screw-press. The discs of platinum thus formed are subsequently heated to whiteness and hammered on an anvil, until a homogeneous welded mass has been obtained.

Although commercial platinum may be prepared by the direct addition of sal-ammoniac to the solution obtained from the ore, the following modification is to be preferred when greater purity is required. The solution, which is generally deep-red, and evolves chlorine from the presence of tetrachloride of palladium, is boiled; whereupon chlorine is expelled, and the palladium present reduced to the state of dichloride.

Chloride of potassium is now added to the solution, which precipitates the platinum as sparingly soluble double chloride of platinum and potassium, leaving the palladium in solution. This precipitate, which, when pure, has a yellow colour, but is red when iridium is present, is collected on a filter, and washed with a dilute solution of chloride of potassium. The double salt of platinum is ignited with twice its weight of potassium carbonate, and the platinum thus reduced to the metallic state, while a portion of the iridium remains as trioxide. The soluble potassium salts are subsequently removed by washing with hot water, and the platinum dissolved by nitro-hydrochloric acid, which leaves the trioxide of iridium undissolved. In order to effect the complete separation of iridium, it may be necessary to repeat, more than once, the precipitation by chloride of potassium and the re-solution of the platinum. The platinum solution thus freed from iridium is treated with sal-ammoniac, and the metal thrown down as double chloride of platinum and ammonium. This is ignited, and the resulting spongy platinum treated as before described.

When platinum ore is attacked by aqua regia, a portion consisting of grains of osmiridium, besides various other substances which have not been removed by washing, always remains undissolved.

DEVILLE AND DEBRAY'S PROCESSES.—Platinum prepared as above described is never quite pure, but contains small quantities of osmium, silicon, &c. In order to remove these impurities, and at the same time to render it more compact and freer from cavities, it is fused in a furnace composed of blocks of well-burnt lime, by means of a hydrogen or coal-gas flame supplied with a current of oxygen.

In a furnace of this description, Deville and Debray succeeded, with a consumption of about 43 cubic feet of oxygen, in melting and refining 25·4 lbs. of platinum in the course of forty-two minutes; very much larger masses have since been treated by this method.

During the operation of fusion, the osmium is expelled in the form of tetroxide, while silicon is removed in the state of silicate of calcium, which, forming a fusible slag, is absorbed by the walls of the furnace. Lime is so bad a conductor of heat that a basin of this substance less than an inch in thickness may be filled with melted platinum without the temperature of the exterior being raised much beyond 150° C.

Deville and Debray likewise introduced the following process for the treatment of platinum ores in the dry way. A small reverberatory furnace, of which the bottom consists of a hemispherical cavity of fire-brick lined with refractory clay, is, after being heated to full redness, charged with a mixture consisting of 2 cwts. of platinum ore and the same weight of galena. The charging occupies some time, as small quantities only are introduced in succession, and the whole is kept constantly stirred until a fusible matte has been produced. A small quantity of powdered glass is used as a flux, and by degrees a weight of litharge, equal to that of the galena employed, is thrown in. The reaction which takes place between the galena and litharge results in the expulsion of sulphur and the reduction of the lead to the metallic state. The reduced lead forms with the platinum a fusible alloy, which is allowed to remain for some time undisturbed in a melted state. In this way the osmiridium, which has not been attacked, and of which the specific gravity is very high, collects at the bottom of the metallic bath. The upper portion of the platiniferous alloy is now drawn off into ingot-moulds, while the residue, containing osmiridium, is added to the next charge. The platiniferous lead is subsequently subjected to cupellation in the ordinary way, and the crude platinum obtained is refined on a bed of lime, by the heat evolved by the combustion of a mixture of coal-gas and oxygen. The platinum thus prepared is nearly pure, is very ductile and malleable, and works well under the hammer.

The inalterability of platinum at high temperatures, together with its power of resisting the action of a great number of the most powerful chemical agents, renders it a useful material for the manufacture of crucibles, evaporating dishes, &c., for laboratory use. Large platinum stills are sometimes employed for the concentration of sulphuric acid; vessels employed for this purpose are strongly gilt on the inside, as unless thus protected, platinum prepared by Wollaston's process soon becomes sufficiently porous to admit of the transudation of the acid. A platinum coinage was introduced some years since in Russia, but not having been found convenient, the coins were ultimately withdrawn from circulation.

ALLOYS OF PLATINUM AND THE ALLIED METALS.—Platinum alloys readily with most of the heavy metals, but few of these compounds have received any very extended practical applications. The so-called platinum bronzes consist of nickel with small quantities (½ to 1 per cent.) of platinum, and 10 to 20 per cent. of tin. An alloy of 13 parts of copper with 3 of platinum is similar in colour to gold. Alloys of gold and platinum in varying proportions are used in Prinsep's Pyrometer for the determination of high temperatures; the melting-point of such alloys increases with the proportion of platinum. The following values are those given by Schertel and Ehrhard, which are based upon Violle's observations:—

| Gold. | Platinum. | Melting-point. |
|---|---|---|
| 100 | 0 | 1075° C |
| 90 | 10 | 1130 |
| 70 | 30 | 1255 |
| 50 | 50 | 1385 |
| 30 | 70 | 1525 |
| 10 | 90 | 1690 |
| 0 | 100 | 1775 |

Crucibles used for the fusion of minerals with caustic alkalies and nitre are sometimes made of gold to which about 5 per cent. of platinum has been added, as they are sensibly harder and less liable to deformation than those made of pure gold, while the resistance to corrosion by the alkaline flux is practically the same.

The most interesting platinum alloys are those with iridium, which are remarkable for their refractory characters, strength and hardness. An alloy of 90 of platinum with 10 of iridium has been made in some quantity by Messrs. Johnson, Matthey & Co., for the International Metric Commission, to be used for the preparation of standard metre bars for the different governments represented on the Commission. The specific gravity of the metal was 21·522 after a first forging, 21·648 after a second forging and rolling, and 21·516 after final drawing to a rectangular section through a series of plates. For standard weights a stronger and denser alloy, containing 20 per cent. of iridium, is considered preferable. This has a specific gravity of 21·614, the volume of a kilogramme being only 46·266, and that of an avoirdupois pound 20·986 cubic centimetres, the latter being equivalent to a cube of about $1\frac{1}{11}$ inch in the side.

The native alloy of osmium and iridium is the hardest and most refractory of the substances associated with platinum. It occurs in crystalline grains, which are practically infusible, are only slowly attacked by the most concentrated aqua regia, and are as rigid as hard tool steel. They are used for the points of gold pens, for which a couple of the larger grains are soldered to the end of the gold nibs, and ground to the required shape on an emery wheel. Latterly a method of casting osmiridium into plates has been introduced by Mr. Holland of Cincinnati, who converts the natural grains into a regulus by fusion with phosphorus, which becomes sufficiently liquid to be poured into a mould formed of two metal plates. The slab of phosphorized metal is refined by exposing it to a very high temperature on a lime cupel in an electric furnace, when the phosphorus is entirely volatilized, leaving the original alloy in a connected mass of the shape of the mould. Osmiridium slabs produced in this way have been used as draw-plates for making gold and silver wires of very exact dimensions, and are said to be preferable to those made of ruby, as the holes, which are made by a diamond drill keep their shape quite as well, and are not so liable to be damaged by chipping.

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